COAL DIVISION

1944

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1944

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FOREWORD

The Coal Division was organized to furnish closer cooperation among those interested in any way in coal, and to promote the advancement of this branch of the work of the Institute through meetings for the reading and discussion of professional papers connected therewith. Unquestionably, in these times of national emergency, with the coal industry facing many difficult problems in keeping the wheels of war industry turning at top speed and the homes and barracks heated, this objective is of utmost importance. During the two regimes represented by this ninth volume of Proceedings, under the able guidance of Chairman Newell G. Alford in 1942 and Chairman Cadwallader Evans in 1943, it seems to the present chairman that the Division has functioned admirably in meeting its obligations.

The papers included in this volume were selected with great care by the Publications Committee, under the chairmanship of John T. Ryan, Jr., and bring into permanent record the latest developments on many technical phases of the industry.

The Coal Industry will face even greater problems in the postwar period. It will need a strong organization of technical men to improve its position technically and fight its battles. Appreciating this, the Division's Membership Committee is engaged in an active campaign, seeking new members, and during the year 1943 gained 153 members. During the last six months of 1943, 14 per cent of the applications to the Institute were from the coal industry.

Past Chairman Cadwallader Evans instituted a Coal Division booklet in 1943, which listed Committee membership and gave their functions. This practice is being continued by the present chairman.

During the present year some reorganization of committees has taken place, with the object of promoting greater ease of functioning and expanding the activities somewhat. A campaign is under way to raise a Scholarship Fund to aid qualified young men, who would otherwise be unable to continue their education beyond the secondary school, to study coal-mining engineering in accredited colleges and universities of their own selection. The far-sighted plan was developed by the Committee on Promotion of Student Interest in Coal Mining and adopted by the Division at its annual meeting in February 1944.

The divisional officers express their appreciation of the whole-hearted support received from committee chairmen and members, authors of professional papers, local committee chairmen for meetings and the many members who have taken an active part in promoting the affairs of the Division.

A. W. Gauger,
Chairman, Coal Division.

State College, Pa.
August 15, 1944.

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Anthracites and Semianthracites in the United States

By ALLEN J. JOHNSON,* MEMBER A.I.M.E.

(New York Meeting, February 1944)

ANTHRACITE coals have been divided into three groups: (1) meta-anthracite, a high-carbon coal that is usually very slow to kindle and difficult to burn, at least on conventional equipment; (2) anthracite, a high-carbon low-volatile coal of widely accepted value as both a domestic and industrial fuel; and (3) semianthracite, a coal lying between anthracite and bituminous coal in carbon and volatile content, and also, generally speaking, in burning characteristics.

ANTHRACITE GROUPINGS

The technical differentiation between these groups, as defined by the A.S.A.-A.S.T.M. Sectional Committee on the Classification of Coals, is as shown in Table 1.

TABLE 1.—*Classification of Coals by Rank*[1]

	Group	Limits of Fixed Carbon, Mineral-matter-free Basis
Anthracitic	1. Meta-anthracite	Dry F.C. 98 per cent or more
	2. Normal anthracite	Dry F.C. 92 per cent or more And less than 98 per cent
	3. Semianthracite	Dry F.C. 86 per cent or more And less than 92 per cent[a]
Bituminous	1 to 5 bituminous coals	Dry F.C. Less than 86 per cent

[a] Nonagglutinating.

Tables 2 and 3 show typical analyses of anthracites and semianthracites and Figs. 1 to 3 show graphically the range of analytical samples. Table 4 shows the commercial sizes of coals.

META-ANTHRACITE AND GRAPHITIC ANTHRACITE

Typical analyses from virtually every field in the United States indicate that there are few if any extensive deposits of meta-anthracite. *Some* of the anthracites of Rhode Island contain almost negligible quantities of volatile and would thus fall into the meta-anthracite grouping on a basis of analyses alone. However, their other characteristics indicate that it is doubtful whether they should be considered as fuels at all, but rather as graphite. In fact, this material is being mined for use as foundry facings and furnace linings.[4] A typical analysis of Fenners Lodge graphite,[4] from Rhode Island, is: graphitic carbon, 55.9 per cent; silica (ash), 40.6; sulphur, 1.6; moisture, 1.9; volatile, 0.

Other Rhode Island coals possess percentages of volatile that would throw them into the normal anthracite rather than the meta-anthracite class. However, this "volatile" is nearly all noncombustible,[3,4] being composed almost entirely of carbon dioxide and inert gases, therefore its definition as volatile is open to discussion, leaving the exact classification of the fuel a moot question. One factor is certain, Rhode Island coal is so different from normal anthracite in both analysis and burning characteristics that it should have a

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Manuscript received at the office of the Institute Dec. 1, 1943.

* Mechanical and Combustion Engineer, Anthracite Industries, Inc., Philadelphia, Pa.

[1] References are at the end of the paper.

TABLE 2.—*Typical Analyses of Anthracites and Semianthracites*

EXCLUSIVE OF PENNSYLVANIA ANTHRACITE

Class	Location			Grouping	Sample		Proximate Analyses, (as Received Basis), Per Cent					B.t.u. (Dry)	Ash Softening Temperature, Deg. F.
	State	County	Town or Mine	Fixed Carbon Moisture-Mineral-Matter-Free	Type[a]	Number	Moisture	Volatile	Fixed Carbon	Ash	Sulphur		
True Anthracites	Rhode Island	Newport		95.0	B	8	17.2	2.9	56.3	23.6	0.4	9,164	
		Providence		97.5	A	4	6.5	2.7	68.4	22.4	0.3	10,346	
	Colorado	Gunnison	Silver Bk.	96.2	B	2	3.0	3.5	87.9	5.6	0.8	14,490	
		Gunnison	Horace	93.7	A	8	2.2	5.6	83.7	8.5	0.7	14,000	2320
		Gunnison	Robinson	92.2	B	1	4.9	7.0	81.8	6.3	0.8	14,160	
		Gunnison	Ruby	96.5	B	1	3.0	3.0	86.5	7.5	0.7	13,920	
		Gunnison	Richardson	93.8	B	1	4.4	5.4	81.9	8.3	0.6	14,000	2410
		Gunnison	Somerset	95.0	B	1	4.2	4.0	76.0	15.8	0.9	12,800	
		Routt	Moffat	92.1	B	1	7.0	6.3	72.2	14.5	0.6	12,420	2920
		Routt	Lionel-Tuck	93.0	B	1	6.7	6.2	80.7	6.4	0.7	13,720	2130
		Routt	Crawford	95.7	B	1	6.9	3.4	75.7	14.0	0.6	12,620	
	New Mexico	Santa Fe	Madrid	93.4	D	100	4.0	5.8	81.6	8.6	0.8	13,520	
		Santa Fe	Madrid	93.2	A	15	3.9	5.8	82.3	8.0	1.1	13,700	2220
Semianthracites	Arkansas	Pope		87.8	B	3	2.4	10.7	77.3	9.6	1.9	13,865	2180
	Colorado	Gunnison	Crested Butte	91.8	D	65	2.3	7.4	80.6	9.7	0.9	13,740	2270
		Gunnison	Genter	91.2	B	4	2.3	8.0	83.0	6.7	0.7	14,290	2220
		Gunnison	Elk Mountain	90.2	B	4	2.7	9.1	82.8	5.4	0.7	14,650	2250
		Gunnison	Raddatz	90.2	B	3	4.1	8.8	80.1	7.0	0.7	14,250	2450
		Routt	Pilot Knob	87.6	A	1	8.4	9.9	69.8	11.9	0.5	12,270	2490
	Pennsylvania	Sullivan	Rock Run	89.0	B	1	2.0	9.6	77.8	10.6	0.6	13,800	2960+
		Sullivan	Bernice	90.7	B	3	2.8	8.3	80.2	9.3	0.6	13,980	2940+
		Sullivan	Forksville	87.0	B	1	1.2	10.1	67.7	21.0	0.5	11,940	2960+
		Northumberland	Trevorton	88.4	B	1	1.7	10.1	76.7	11.5	0.8	13,630	2940+
	Virginia	Pulaski	Pulaski	87.8	B	2	4.7	9.8	71.2	14.3	0.6	13,080	2930
		Montgomery	Blakesburg	88.0	D	4	4.2	9.3	68.2	18.3	0.6	12,480	
		Montgomery	Merrimac	86.0	B	3	2.1	11.4	67.3	19.2	0.6	12,220	
		Wythe	Gunton	86.4	B	1	5.2	10.3	65.1	19.4	1.2	12,210	
	Washington	Whatcom	Glacier	90.7	B	4	5.2	7.8	77.0	11.6	1.0	13,600	2630

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[a] A indicates Bureau Mines Mine Sample; B, Geological Survey Mine Sample; D, Delivered Coal.

TABLE 2.—(*Continued*)

AVERAGE ANALYSES OF VARIOUS COALS BY STATES (EXCEPT PENNSYLVANIA)

Location		Grouping		Proximate Analyses, Per Cent					
State	Type Coal	Fixed Carbon, Moisture-Mineral-Matter Free	Number of Samples	Moisture	Volatile	Fixed Carbon	Ash	Sulphur	Heat Value, B.t.u. (Dry)	Ash Softening Temperature, Deg. °F.
Rhode Island	"Graphitic" anthracite	96.8	12	11.8	2.8	62.4	23.0	0.3	9,958	2450
Colorado	True anthracite	94.2	17	4.7	4.9	80.8	9.6	0.7	13,570	2220
New Mexico	True anthracite	93.3	115	3.9	5.8	81.9	8.3	1.0	13,610	2180
Arkansas	Semianthracite	87.8	3	2.4	10.7	77.3	9.6	1.9	13,865	2340
Colorado	Semianthracite	90.2	77	4.0	8.6	79.3	8.1	0.7	13,840	2950+
Pennsylvania	Semianthracite	88.8	6	1.9	9.5	75.6	13.1	0.6	13,340	2930
Virginia	Semianthracite	86.8	10	4.3	10.2	67.9	17.8	0.7	12,500	
Washington	Semianthracite	90.7	4	5.2	7.8	77.0	11.6	1.0	13,600	2630

separate classification—possibly as graphitic anthracite.

Narragansett Field

The Narragansett field of Rhode Island and Massachusetts[3,4] is a basin of 1000 square miles, roughly including the part of Rhode Island that is east of Providence as well as the southeastern part of Massachusetts south of Brockton and east of the Massachusetts-Rhode Island state line.

The coal is extremely variable in character and quality, ranging from anthracite to graphite. However, virtually all of this coal possesses common undesirable characteristics that render it unsuitable for use on domestic equipment or with natural draft. Because of these, special handling, specific equipment, and undercover storage are practically requisites.

These coal beds are neither uniform in thickness nor continuous. Compression has resulted in squeezing the plastic coal into more or less lense-like masses. As a result, the coal is so sheared and fractured that it possesses little coherence and is extremely friable, thus yielding a very large percentage of the smaller sizes. As most of the run of mine coal is of the small "birds-eye" size this fact alone mitigates against its use on natural draft.

The fracturing of the coal has permitted the percolation of mineral bearing solutions which have deposited a material similar to asbestos or asbestos-quartz throughout the coal, thus materially increasing the percentage of non-combustible and rendering ignition extremely difficult.[3]

Coal from the Narragansett field is extremely high in moisture content when first mined and, in fact, is hydroscopic, especially in the smaller sizes. As much as 15 per cent moisture will be absorbed by dry coal in a relatively short exposure to moist air. As a result, the B.t.u. value is largely dependent upon the conditions of mining and storage. Air-dried coal will reach 10,000 or even 11,000 B.t.u. but the same coal may drop to 6000 to 8000 B.t.u.

TABLE 3.—*Analyses of Pennsylvania Anthracite*[a]

Coal	Moisture, Per Cent	Proximate Analyses, Per Cent									Heat Value, B.t.u.			Ash Softening Temperature, Deg. F.
		As Received				Dry Coal Basis								
		Volatile	Fixed Carbon	Ash	Sulphur	Volatile	Fixed Carbon	Ash	Sulphur	Fixed Carbon, Moisture-Mineral-Matter-Free	As Received	Dry	Moisture and Ash-Free	
Egg	4.2	4.3	82.6	8.9	0.7	4.4	86.4	9.2	0.7	95.2	13,100	13,670	15,060	2890
Stove	4.3	4.3	82.5	8.9	0.7	4.4	86.3	9.3	0.7	95.2	13,040	13,630	15,030	2890
Chestnut	4.8	4.2	82.1	8.9	0.7	4.3	86.0	9.7	0.7	95.2	12,900	13,550	15,010	2890
Pea	6.2	4.1	79.1	10.6	0.7	4.3	84.4	11.3	0.7	95.1	12,470	13,290	14,980	2890
Buckwheat	6.0	4.2	78.8	11.0	0.7	4.6	83.7	11.7	0.7	95.0	12,430	13,220	14,970	2880
Rice	5.7	4.2	78.4	11.7	0.7	4.5	83.1	12.4	0.7	94.9	12,330	13,080	14,030	2900
Barley	7.2	4.1	76.1	12.6	0.7	4.4	82.0	13.6	0.7	94.9	11,960	12,890	14,920	2880
Averages	5.5	4.[illegible]	79.9	10.4	0.7	4.4	84.5	11.0	0.7	95.1	12,600	13,330	14,970	2890

[a] Based upon 268 analyses of samples from 41 representative breakers located throughout entire anthracite region of Pennsylvania. Reported by U. S. Bureau of Mines in R. I. 3283.

upon exposure to moisture. As large lumps do not dry out quickly, the smaller sizes usually may be considered to have the higher heat values.

TABLE 4.—*Commercial Sizing of Coals*
INCHES

Size	Anthracites		Semianthracites	
			Arkansas	
	Pennsylvania	New Mexico	Bernice	Sparda
Block				14–7
Grate and Broken	4⅜–3¼ (Broken)	4–2½	7½–4¼	7–5 7–4
Furnace			7½–2¼	
Egg	3¼–2 7/16	2½–2¼	4¼–2½	5–2¼ 4–2¼
Stove	2 7/16–1⅝		2¼–½	2¼–1½
Range			2¼–⅝	
Base burner		1½–⅞		
Nut			1¼–⅞	1½–⅞ 1⅛–⅝
Chestnut	1⅝–1 3/16	1½–½	1–⅝	1–⅞
Pea	1 3/16–9/16	⅞–½	⅞–⅝	⅞–⅝
Buckwheat	9/16–5/16	½–5/16		
Buckwheat No. 2 (rice)	5/16–3/16			
Buckwheat No. 3 (barley)	3/16–3/32			
Buckwheat No. 4	3/32–3/64			
Duff		5/16–0		
Slack				⅞–0 ⅝–0

Additional disadvantages include "explosive" burning due to entrained moisture, clinkering, and difficulty in washing with existing equipment, owing to high specific gravity (average of 8 samples = 2.00 sp.gr.).

One ton of this coal occupies 25 to 30 cu. ft. as compared with 35 to 40 for normal anthracite, thus leading to frequent consumer complaint of "short weight."

In spite of these manifold disadvantages, numerous attempts have been made to mine and market this coal for more than a century. Both coal and graphite have been mined at Portsmouth and near Cranston, Rhode Island. In Massachusetts, coal was once mined at Mansfield, but the workings have been filled with water since 1838 and little authoritative information is available.

From 1860 to 1883, the Faunton Copper Co. used as much as 10,000 tons per year successfully to smelt copper ore from Cuba and South America. This project was abandoned only when the copper industry moved west.

In 1887 the New York Carbon Iron Co. shipped 250 to 300 tons of Cranston coal each week to Pittsburgh, for making billets for blooms.

Even briquetting was attempted but failed because of the low grade and high ash of the coal.

In the past 60 years, only sporadic attempts have been made to mine Rhode Island coal for domestic consumption.

A report to the Massachusetts Legislature* summarizes the advantages and disadvantages of Narragansett field anthracite as follows:

Disadvantages: (1) irregular form and thickness of the coal beds; (2) intimate fracturing and friability of the coal; (3) large percentage of fine material in mining (main mine output would be smaller sizes); (4) high content of noncombustible material or ash; (5) high moisture content; (6) low heating value; (7) difficulty of combustion; (8) continual attendance necessary in burning it.

Advantages.—Location in a region that is some distance from the nearest available fuel; hence low cost of delivery.

SEMIANTHRACITE

Semianthracite frequently is found as a part of veins or measures having a general bituminous or anthracite grouping. However, the principal deposits of semianthracite in the United States are in Arkansas, Pennsylvania (Sullivan County), Virginia and Washington.

Reserves and Production

Reserves.—Fig. 4 shows the estimated reserves and approximate annual produc-

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* By W. F. Jones, Assistant Professor, Massachusetts Institute of Technology (included in reference 3).

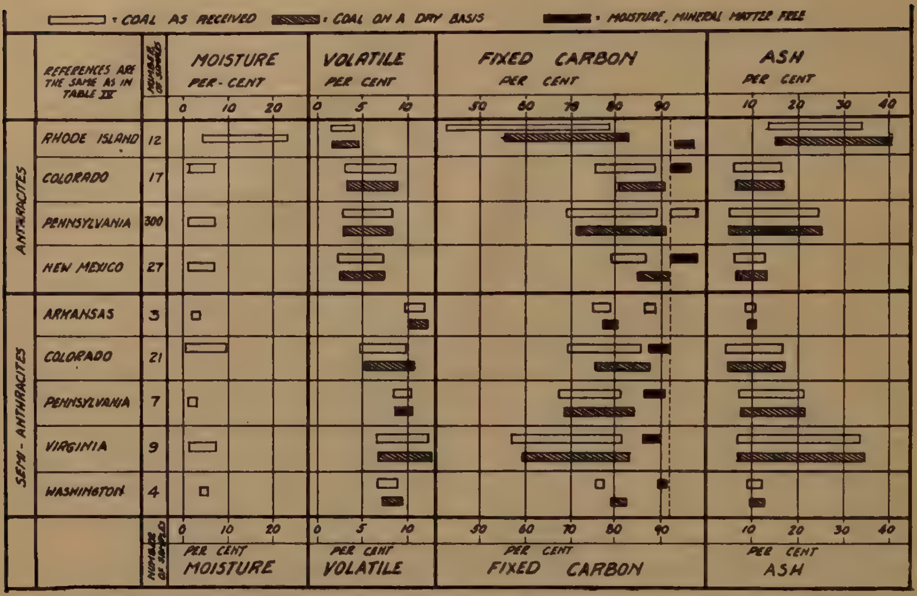


FIG. 1.—RANGE OF ANALYTICAL VALUES OF ANTHRACITES AND SEMIANTHRACITES.

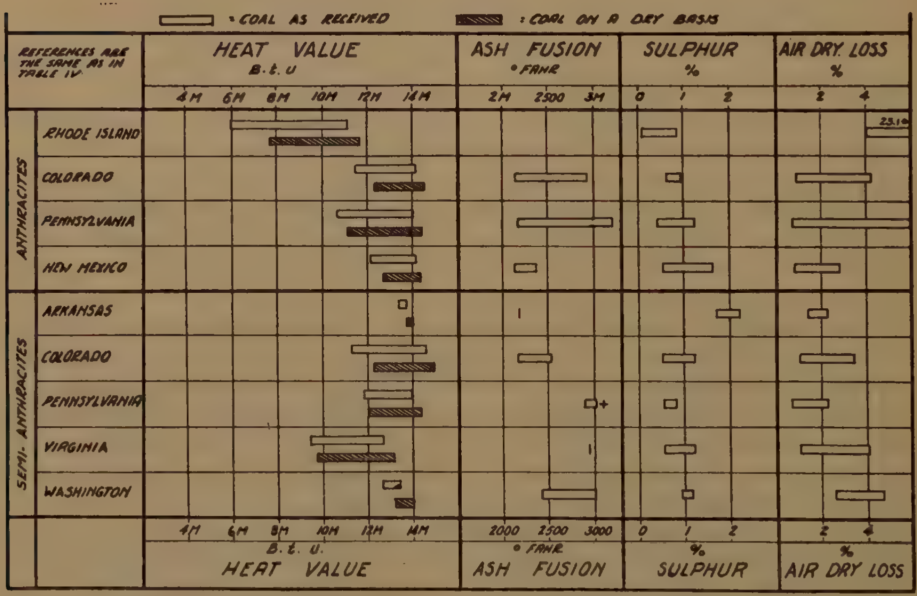


FIG. 2.—RANGE OF ANALYTICAL VALUES OF ANTHRACITES AND SEMIANTHRACITES.

tion of semianthracite in these several states.

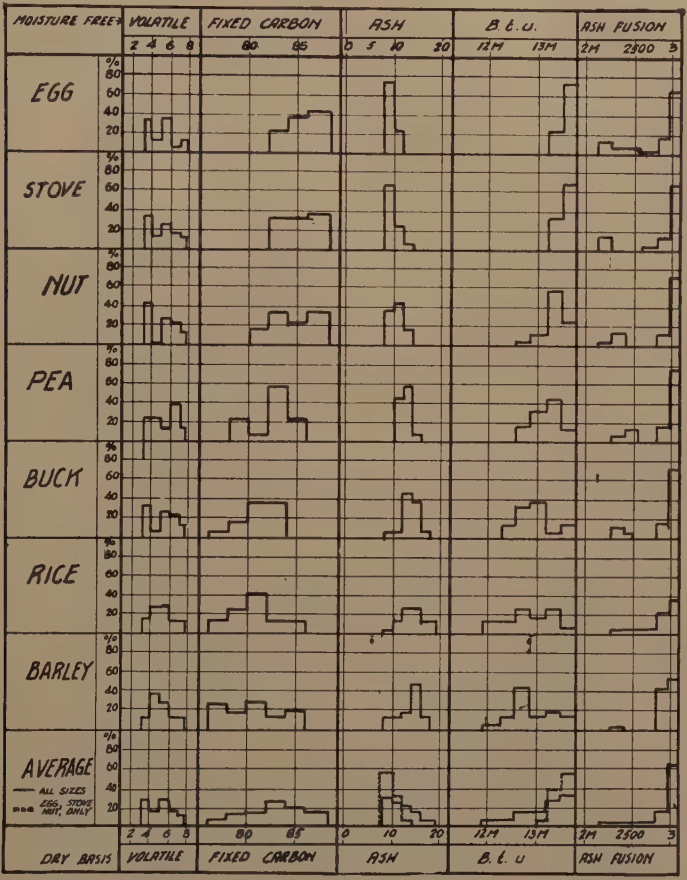


FIG. 3.—RANGE OF ANALYSES OF PENNSYLVANIA ANTHRACITE, 128 BREAKER SAMPLES.

Arkansas.—Relatively extensive deposits of semianthracite are in Pope and Johnson Counties, in western Arkansas. Production represents about 15 per cent of the state's output of coal.

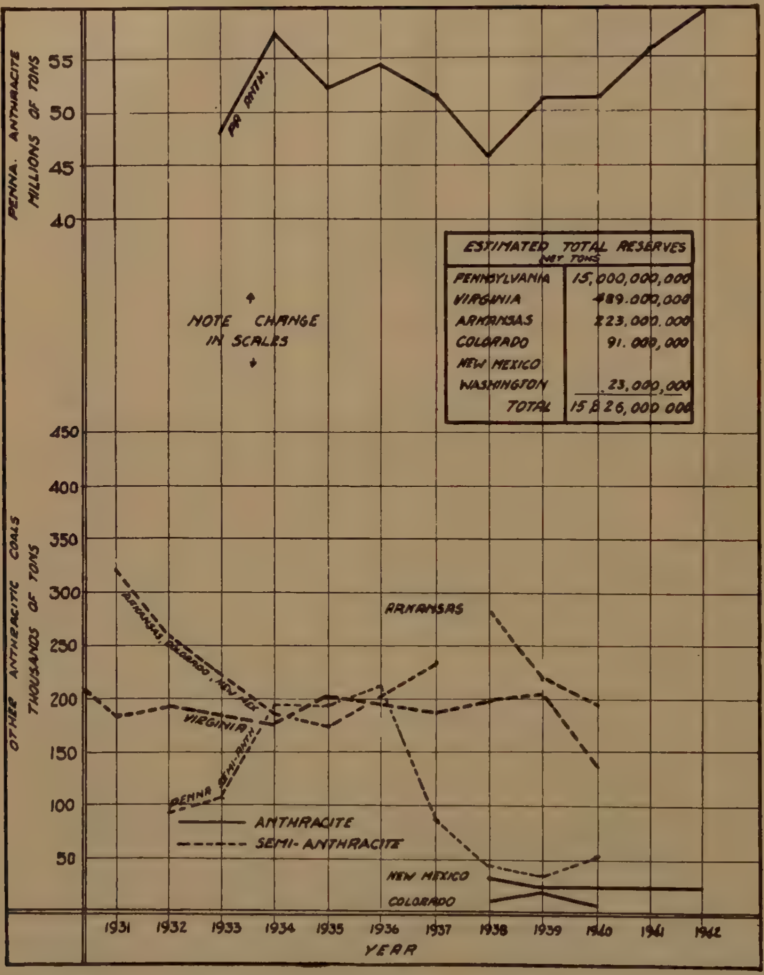
Although the Arkansas Geological Survey divides this field into four districts, the entire area usually is referred to as the Sparda field,[10] subdivided into the Bernice or Russelville field in southeast Pope County and the Sparda bed in the northwest (Johnson County).

The coal beds are well exposed in this region and have suffered considerable folding, faulting, and erosion.[11]

The principal seam mined, the Hartsborne, has coals of three different ranks in different localities—semianthracite, low-volatile and medium-volatile bituminous.

Because this coal is the hardest in structure and lowest in volatile found in this region, it is frequently called "Arkansas anthracite," although the analyses clearly indicate that it is semianthracite in rank (Table 2).

As frequently is characteristic of semianthracite, the ash-fusion temperature is low, the average being about 2180°F. The sulphur content is also high as compared with other anthracitic coals (1.7 to 2.2 per cent).



ESTIMATED TOTAL RESERVES NET TONS	
PENNSYLVANIA	15,000,000,000
VIRGINIA	489.000,000
ARKANSAS	223,000.000
COLORADO	91.000,000
NEW MEXICO	
WASHINGTON	23,000,000
TOTAL	15 826,000 000

FIG. 4.—PRODUCTION AND RESERVES OF ANTHRACITES AND SEMIANTHRACITES IN THE UNITED STATES.[12–16]
Includes all fuel mined, net tons.

In addition, the excessive folding of the region would be expected to have produced a coal of high friability and large percentages of small sizes. Otherwise, the coal is of good analysis and quality.

Since the inauguration of an effective smoke prevention campaign in St. Louis, this fuel has been in considerable demand, as the region is operating at capacity production.

Pennsylvania.—In Pennsylvania semianthracite is found in Sullivan County and the western part of Wyoming County. Mines are at Bernice (Connell), Laporte (Rock Run), and Forksville (Monahan), Sullivan County.

Some coal at the extreme western tips of the Pennsylvania anthracite region's western middle fields (Northumberland

County) and southern field (Dauphin County) analyze as semianthracites but only two small mines have been located in these tips and there are none in Wyoming County.

The principal practical differences between the physical properties of Pennsylvania anthracite and semianthracite are a higher volatile content with a tar deposit after combustion, somewhat more distinct laminations due to luster contrasts, more of a tendency toward a rectangular rather than conchoidal fracture, lower specific gravity (below 1.47), somewhat softer body (2.50 to 2.75 as against 2.75 to 3.00), and more friable (50 per cent as against 35 per cent).

While these differences combine to form a coal that is distinctly inferior to true anthracite, they are unfortunately difficult to detect by any means other than through actual usage or an analysis and determination of the moisture and mineral-matter-free percentages of fixed carbon.

Colorado.—In Colorado, both anthracite and semianthracite are mined in the Crested Butte field of Gunnison County. There is also a small isolated deposit of semianthracite in the Green River bituminous field of Routt County, northwest of the town of Steamboat Springs.

Semianthracite mined in this field is normal in all characteristics, including a low ash-softening temperature range of between 2150° and 2530°F. with a corresponding tendency to clinker.

Virginia.—Semianthracite is mined extensively in the Brushy Mountain field of Pulaski, Wythe, and Montgomery Counties in Virginia. Although small (3 per cent) as compared with the Pennsylvania anthracite reserves, the estimated half a billion tons of semianthracite in Virginia nevertheless forms the second largest reserve of anthracitic coals in the United States. Furthermore, because of proximity to Pennsylvania anthracite markets, this coal frequently is sold in competition where the common trade name "Virginia anthracite" often leads to the impression that it is anthracite rather than semianthracite.

However, by analysis there is no doubt concerning the accuracy of its classification as semianthracite. The highest moisture-mineral-matter-free fixed carbon of any sample reported by the Bureau of Mines[7] was 90 per cent—two full per cent below the 92 per cent borderline between anthracite and semianthracite. The volatile analyzes as high as 12.6 per cent, with an average of 10.2 per cent, and the ash as high as 33.8 per cent (delivered sample) with a 17.8 per cent average. Thus clearly this is not a borderline coal.

In one series of comparative tests,* Virginia semianthracite was found to have a use value approximately 88 per cent of Pennsylvania anthracite. As characteristic of semianthracite, it burned much more rapidly, with greater difficulty of control. Considerable soot was emitted, thus adding to the inconvenience of the higher ash content.

West Virginia.—The U. S. Geological Survey map, Coal Fields of the United States, also shows deposits of anthracitic coal west of Harrisonburg in Buckingham and Augusta Counties, and in Berkley County, West Virginia (west of Martinsburg). No further data could be found by the author.

Washington.—Semianthracite in the state of Washington lies in the extreme northwestern county (Whatcom) on a ridge south of Glacier Creek and northwest of Mt. Baker.

The thickness of veins extends to 14 ft. or more but much of it is in the form of finely disintegrated crushed and weathered fragments, which appear to be soft. Impure laminated coal also usually forms a substantial part of the veins.

No production on a commercial scale has yet taken place and recent work has not developed beyond the prospective stage, because of these adverse properties and because of the lack of continuity and faults in the region.

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* Anthracite Industries Laboratory Report 2212.

True Anthracites

While scattered analyses of true anthracites sometimes are found in regions largely devoted to the production of semianthracite, and even bituminous, it is certain that the only commercial production of true anthracite is in the states of Colorado, Pennsylvania, and New Mexico, exclusive of the graphitic anthracite of Rhode Island.

Of known reserves (Fig. 4), Pennsylvania has over 98 per cent. In production, Pennsylvania as of 1940 shipped all but about 50,000 tons of true anthracite and all but 400,000 tons of all anthracitic coals in the United States.

For geographical reasons, the anthracites of Colorado and New Mexico are not competitive with Pennsylvania anthracite. Virginia semianthracite has been in competition as far north as Philadelphia. Pennsylvania anthracite and semianthracite are competitive, as are Pennsylvania anthracite and Arkansas anthracite in St. Louis—although St. Louis can hardly be called a logical marketing area for Pennsylvania anthracite.

Pennsylvania.—Pennsylvania anthracite occurs in one region of approximately 500 square miles in northeastern Pennsylvania. Four fields are recognized; the Northern field, Lackawanna and Luzerne Counties; the Eastern Middle field, Luzerne and Carbon Counties; the Western Middle field, Northumberland, Columbia, and Schuylkill Counties; and the Southern field, Schuylkill and Dauphin Counties.

With the possible exception of the practically unmined extreme western tips of the Southern field, all coal mined in this region is true anthracite.

Turner[2] has noted and plotted isovolatile lines for the entire region, separating areas of coals having similar analyses. There is some corresponding variation in burning characteristics, paralleling these lines; however, considering the area of the field, this variation is remarkably slight, all coal being of extremely uniform quality. In fact, in actual combustion the principal difference between any two Pennsylvania anthracites of equal preparation is in the adjustment of drafts to correspond to the burning index. In most cases even this will not be noticeable to the consumer. Preparation and sizing also have been standardized throughout the region.

Considerable material has been published covering virtually all phases of the combustion and use of Pennsylvania anthracite and for the sake of brevity, this will not be repeated. The four Lehigh Conference *Transactions* and various Anthracite Industries reports and publications are recommended for further information and data.

Colorado.—Colorado anthracite and semianthracite are found in the same fields. The analyses indicate a very satisfactory coal, with low to medium volatile, low ash, and high heat value.

The unfavorable points are a relatively low ash-softening temperature, with corresponding tendency to clinker; and the fact that both anthracite and semianthracites are found in the same location, which might lead to a variation in the quality of fuel obtainable from a given section.

New Mexico.—New Mexican anthracite is mined in the small Cerrillos field of Santa Fe County. The coal is of good quality and excellent analyses, with 5.8 per cent volatile and 8.3 per cent ash. The ash-softening temperature is low

Distribution is largely by rail to Arizona, California, Colorado, Kansas, Nebraska, New Mexico, Texas, and Wyoming.

In recent years production has been materially held down by extensions of natural-gas pipe lines in the territory served. However, it is reported that with

O.P.A. ceilings as high as $9.05 the coal is at present moving on freight rates as high as $9.00 for a total price on sidings of $18.05.

CONCLUSIONS

Anthracitic coals are known to occur in eight states—Arkansas, Colorado, Pennsylvania, Massachusetts, New Mexico, Rhode Island, Virginia and Washington—and there is a possibility of isolated uncommercial deposits as a part of seams of other ranks in other states.

As coals cover a continuous range of analyses even in the same locality, any dividing line between anthracite and semianthracite must necessarily be arbitrary. This has been set at 92 per cent moisture-mineral-matter-free fixed carbon, not only because it seemed to be a logical division from a point of customary usage and nomenclature, but also, and more important, because coals below 92 per cent give a tar deposit upon combustion and coals above 92 per cent do not. Furthermore, semianthracites, by reason of higher volatile, burn faster and usually, although not always, clinker more readily. Therefore, the distinction between anthracite and semianthracite is an important one, especially in areas where both are obtainable. Nevertheless, borderline cases should be recognized, particularly in the coals of Colorado, the individual fuels being judged for their actual burning characteristics with principal reference to the production of tar, soot, and clinker.

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The Pittsburgh Coal Seam in Pennsylvania—Its Reserves, Qualities and Beneficiation

By David H. Davis* and John Griffen,† Members A.I.M.E.

(New York Meeting, February 1944)

Much of the ground to be covered by this paper was ably covered by a paper presented by Messrs. Morrow and Jordan[1] before a joint meeting of the Iron and Steel Section of the Engineers Society of Western Pennsylvania and the A.I.M.E. Pittsburgh Section at Pittsburgh in the spring of 1940. As that paper was not published, the present authors have availed themselves of Mr. Morrow's kind permission to use information given therein. Similar figures for the latest years available have been added to many statistics originally presented. It is unfortunate that Mr. Jordan's death removes his stimulating guidance and help, which the authors would have welcomed and have found invaluable in the past.

One interested in the Pittsburgh seam should not fail to read the exhaustive and authoritative paper by Eavenson[2] presented in 1938.

As Allegheny, Fayette, Greene, Washington, and Westmoreland Counties contain practically all the reserves and furnish nearly all of the production from the Pittsburgh seam in Pennsylvania at the present time, this area only is intended when reference is made to the Pittsburgh seam in Pennsylvania.

On much of the seam in Greene and Washington Counties, which contain a large proportion of the total reserves, insufficient data are available to determine the detailed characteristics of the seam with any degree of finality. It is hoped that the data presented here will fairly well indicate the probable characteristics.

Economic Importance of the Pittsburgh Seam

The Pittsburgh seam in Pennsylvania is of primary importance as a coal resource, not only because of its substantial reserves but because it contributes such a large proportion of the coal used in the country and an even larger proportion of that used in the manufacture of coke. Mr. Ashley[3] placed the estimated recoverable reserves (1935) of the Pittsburgh coal bed in Pennsylvania at 7,500,000,000 tons.* Although a considerable portion of this tonnage is not now suitable for coking coal, because of its high sulphur content, undoubtedly the supply of coking coal will last for many years and that not suitable for coking will furnish industry with an ample supply of energy.

In attempting to assess the contribution of the Pittsburgh seam to total coal production and to that of coal used in making coke, one finds that the statistics published annually by the U. S. Bureau of Mines, Minerals Yearbook, do not give tonnages classified as to seams but only as to districts or states and counties. However, it happens that, at present and during much of the

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[1] References are at the end of the paper.

* Throughout this paper, "tons" will be used to denote net tons of 2000 pounds.

past, the Pittsburgh seam has contributed about 95 per cent of the output of the five counties of Pennsylvania here considered. It is therefore believed that the estimates of tonnage "probably from the Pittsburgh seam" given in Tables 1, 2, 3 and 4 are substantially accurate. Tables 1 to 4 show that the Pittsburgh seam in recent years has made the following contributions to the economy of the United States:

1. It has supplied from 12 to 15 per cent of the total tonnage and 15 per cent of the total value (at the mines) of bituminous coal produced.

2. It has furnished 32 to 37 per cent of the tonnage of coal used in the manufacture of by-product coke.

3. It has supplied 50 to 85 per cent of the tonnage of coal used in the manufacture of beehive coke.

Table 2 shows the extensive area (13 states) to which this coal is shipped for the manufacture of by-product coke. Coal from this seam made into beehive coke is used entirely within Pennsylvania, the beehive ovens being invariably at the mines or closely adjacent thereto.

A coal bed as extensive as the Pittsburgh seam (2077 square miles in Pennsylvania[4]) varies considerably, but coal from the Pittsburgh seam in Pennsylvania invariably makes coke that is usable in the blast furnace. There may be variations in the yield of coke and in its physical properties, which affect its value in the blast furnace, but these are usually of relatively minor importance.

Variations in the chemical constituents of the coke such as ash, sulphur and phosphorus, due to variations in the analysis of the coal, may be more detrimental, and this is particularly true of sulphur. It will be shown later on that the sulphur analysis of the Pittsburgh-seam coal in Pennsylvania varies very widely, and much of the sulphur is so intimately associated with the coal substance that its

TABLE 1.—*Production and Value at the Mines of Bituminous Coal Only*[a]

Region	1937[b]	1938[b]	1939		1940		1941	
	Production, Net Tons	Production, Net Tons	Production, Net Tons	Value, $1000	Production, Net Tons	Value, $1000	Production, Net Tons	Value, $1000
Counties in Pennsylvania								
Allegheny	15,218,351	11,988,561	12,922,910	24,424	15,676,475	30,256	16,135,666	35,498
Fayette	20,820,042	10,095,831	15,672,679	34,637	22,191,047	46,601	22,793,166	53,336
Greene	4,607,642	3,474,274	3,939,164	8,706	3,202,159	11,237	6,132,243	13,430
Washington	18,895,224	12,595,645	15,484,985	30,041	19,395,154	38,596	20,097,652	45,220
Westmoreland	10,674,423	6,169,524	7,329,434	13,559	8,985,347	16,982	10,930,010	23,827
Total	70,215,682	44,323,835	55,349,172	111,367	71,450,182	143,672	76,088,737	171,311
Probably from Pittsburgh seam	66,704,900	42,107,600	52,581,700	106,136	67,877,700	136,778	72,284,300	162,941
Total U.S.A.	442,313,030	345,546,843	391,877,279	723,081	457,832,299	876,676	511,373,413	1,122,553
Probably from Pittsburgh seam as percentage of total U.S.A.	15.1	12.2	13.4	14.7	14.8	15.6	14.1	14.5

[a] Minerals Yearbook, U. S. Bur. Mines.
[b] Data on value of coal not given for 1937 and 1938.

elimination and control are a challenge to engineering skill.

TABLE 2.—*Coal from Western Pennsylvania Purchased for Manufacture of By-product Coke in the United States*[a]

NET TONS

Field Where Coal Was Produced	Probable Seam	1937	1938	1939	1940	1941	1942
Connellsville	Pittsburgh	15,167,424	7,681,777	13,084,727	17,868,048	16,306,546	17,069,335
Freeport	Freeport	1,555,926	1,267,950	1,354,891	1,982,018	1,877,533	1,629,956
Pittsburgh	Pittsburgh	9,154,109	6,375,832	8,675,795	9,627,541	11,867,295	12,559,474
Somerset	Unknown	617,358	266,965	471,923	662,302	552,272	554,991
Westmoreland	Pittsburgh	1,364,765	468,591	485,505	915,380	1,022,500	438,866
Total western Pennsylvania		27,859,582	16,061,115	24,072,841	31,055,289	31,626,146	32,252,622
Total probably from Pittsburgh seam		25,686,298	14,526,200	22,246,027	28,410,969	29,196,341	30,067.675
Total coal purchased in U.S.A.		70,177,977	45,843,671	62,062,052	77,343,243	83,187,302	90,401,393
Coal probably from Pittsburgh seam as percentage of total U.S.A.		36.6	31.7	35.8	36.7	35.1	33.3

Field Where Coal Was Produced	States Where Coal Was Consumed, In Order of Importance
Connellsville	Pennsylvania, Ohio, West Virginia, New York, Illinois, Minnesota, Michigan, New Jersey, Indiana, and Connecticut.
Pittsburgh	Pennsylvania, New York, Ohio, Michigan, Illinois, Wisconsin, Minnesota, New Jersey, Connecticut, Massachusetts, and West Virginia.
Westmoreland	Pennsylvania, Maryland, New York, Minnesota, Ohio, Connecticut, and Wisconsin

[a] Minerals Yearbook, U. S. Bur. Mines.

TABLE 3.—*Coal Used in Beehive Ovens in Pennsylvania, by Districts*[a]

District	Probable Seam	Coal Used, Net Tons					
		1937	1938	1939	1940	1941	1942
Allegheny Mountain and Allegheny Valley	Unknown	64,103	38,777	43,886	37,829	92,699	Detailed figures not available
Connellsville	Pittsburgh	1,355,627	167,366	685,777	1,557,775	4,631,345	
Lower Connellsville	Pittsburgh	1,659,105	248,202	622,631	1,619,619	2,959,236	
Upper Connellsville	Pittsburgh	215,112	65,003	47,685	118,484	412,293	
Pittsburgh and other districts[b]	Mostly Pittsburgh	612,807	245,905	375,328	663,605	1,100,790	
Total		3,906,754	765,253	1,775,307	3,997,312	9,196,363	11,292,336
Total probably from Pittsburgh seam		3,742,000	686,000	1,681,000	3,860,000	8,900,000	11,000,000
Total U.S.A.		4,926,824	1,359,876	2,297,785	4,802,996	10,529,316	12,875,812
Coal probably from Pittsburgh seam as percentage of total U.S.A.		76.0	50.4	73.2	80.4	84.5	85.4

[a] Minerals Yearbook, U. S. Bur. Mines.
[b] Includes Bedford and parts of Indiana and Westmoreland Counties.

The rapid rate at which the reserves of high-grade Pittsburgh-seam coal of established coking quality in Pennsylvania are being depleted make it important to examine the present situation and assess the future. On the one hand is the problem of exhaustion of the better coals and on the other the more difficult requirements to meet in the manufacture of steel and alloy-steel products. This latter requirement, it is generally agreed, must depend on better, and particularly more uniform, raw materials for satisfactory costs. King[5] says:

It is generally recognized that undoubtedly deterioration of Pittsburgh coking coals will be accelerated by the depletion of some of the better grades of coal; the extension of mine mechanization, with attendant degradation of coal quality; and the opening of new coal fields, some of which are known to be fairly high in sulphur.

No exact and complete specifications have ever been accepted for coking coals. Rose[6] aptly states that coal selection is a matter of choosing the coal or coals that appear to offer the most advantages. He gives some of the generally accepted criteria for selection as follows:

TABLE 4.—*Pittsburgh-seam Coal Used in Manufacture of By-product and Beehive Coke*
NET TONS, BASED ON DATA IN TABLES 2 AND 3[a]

Coal Probably from Pittsburgh Seam Used for	1937	1938	1939	1940	1941	1942
By-product coke	25,686,298	14,526,200	22,246,027	28,410,969	29,196,341	30,067,675
Beehive coke	3,742,000	686,000	1,681,000	3,860,000	8,900,000	11,000,000
Total	29,428,298	15,212,200	23,927,027	32,270,969	38,096,341	41,067,675
Total coal used in manufacture of by-product and beehive coke in U.S.A.	75,104,801	47,203,547	64,359,837	82,146,239	93,716,618	103,277,205
Coal probably from Pittsburgh seam as percentage of total U.S.A.	39.2	32.2	37.2	39.3	40.7	39.7

[a] Minerals Yearbook, U. S. Bur. Mines.

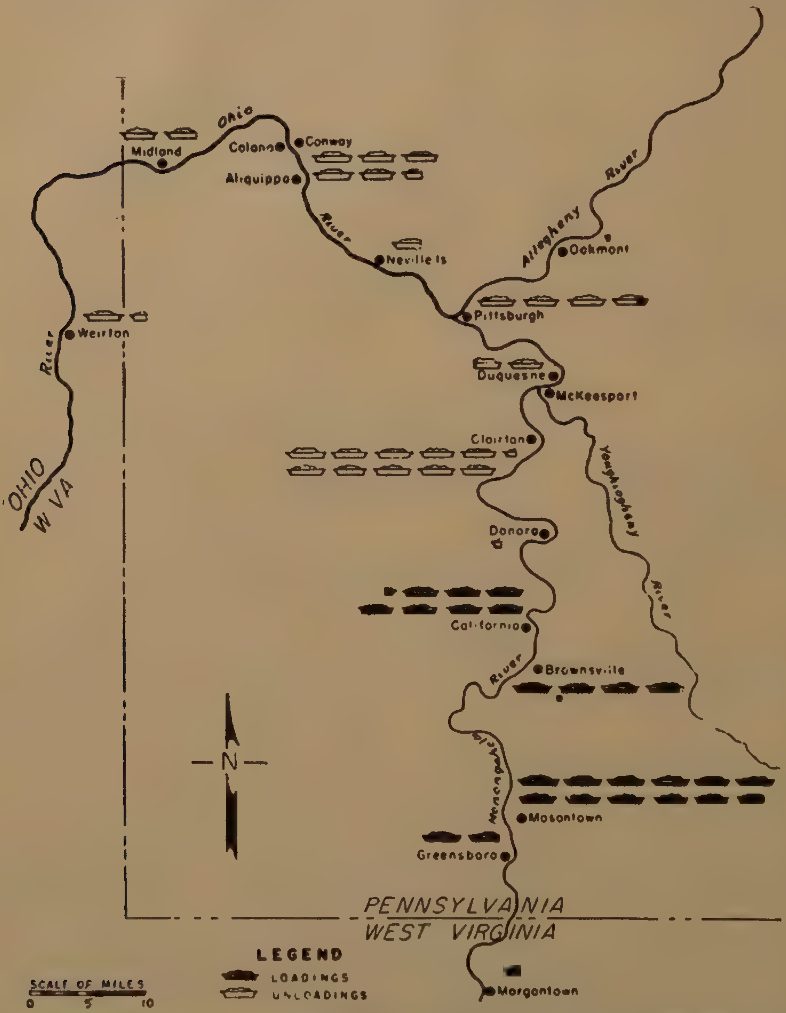
> The best cokes contain less than 10 per cent of ash, and a strong prejudice exists against coke having more than about 12 per cent, if better coke is available. This means that the coals used should contain not over 9 per cent ash, which is the standard specifications of the American Society for Testing Materials for both gas and coking coals.
>
> The permissible ash and sulphur content of blast-furnace coke in each locality depends upon the purity of the available coals. Freight rates are in some cases a determining factor. A plant getting coal by water transportation might find it advisable to use a coal less satisfactory than what might be available by rail.
>
> In the Pittsburgh district, large quantities of crushed run-of-mine Pittsburgh seam coal having as much as 10 per cent ash are coked in by-product ovens . . . Shale and slate have an injurious effect on coke size and strength, the full extent of which is probably not generally realized.

Sulphur.—The A.S.T.M. specifications previously quoted stated that coking coal shall be of such composition that the coke produced shall contain not more than 1.3 per cent sulphur for blast-furnace coke. Less than 1 per cent sulphur is preferable, and coke having as much as 1.50 to 1.75 per cent sulphur is not considered acceptable at many furnaces. The percentage of sulphur eliminated during the coking process depends upon the coals used, but will average around 40 per cent for coals giving 70 to 75 per cent coke yield. In a general way this means that the percentage of sulphur in coke is from 0.01 to 0.3 less than in coal from which it was made, but the sulphur content of coke is sometimes as high or even higher than the coal used.

Phosphorus.—The iron used in both the acid open-hearth and bessemer processes must be very low in phosphorus, and coke used in making such iron should preferably contain less than 0.010 per cent of this element. All of the phosphorus in the coal passes into the coke, and all of the phosphorus in the materials charged into a blast furnace passes into the iron. A coal containing as much as 0.020 per cent phosphorus would generally be considered of limited value, as most American blast-furnace cokes are very low in phosphorus.

The phosphorus content of coke used in making basic iron is of little importance as the basic steelmaking process readily removes this impurity.

It may be noted that no change has been made in the A.S.T.M. specifications for gas and coking coals since Rose's paper appeared, for the latest published are those adopted in 1924, from which Rose quoted.



SOURCE: Board of Engineers for Rivers and Harbors, War Dept

FIG. 1.—DISTRIBUTION OF BITUMINOUS COAL ORIGINATING ON THE MONONGAHELA RIVER, 1941. (*Coal Age*, October 1943.)

If exact and complete specifications were written for coking coal, each user probably would find certain conditions that would force him to ignore or compromise on certain specifications. The freight rates might be prohibitive or he might have a very decisive advantage in transporting a somewhat inferior coal cheaply by water, which he could not afford to lose. He might own a mine of which he could not well dispose. All such factors force a compromise as to specifications of a coking coal and each user by means of actual trials in coke plants and blast furnaces has attempted to select the coals best suited to his uses and location. In some cases, he has found that by mixing coals, even where their analyses are apparently identical, coke of more suitable character is obtained.

In the recent past, about 1.20 per cent sulphur has been considered the upper limit for Pittsburgh-seam coking coal, but this limit has not been rigidly applied. Where cheap water transportation plays a dominant role, an upper limit of 1.50 per cent sulphur, or perhaps slightly more, might be considered equivalent to a 1.20 per cent sulphur limit for coal transported by rail.

That river transportation plays a very important role in the mining and distribution of Pittsburgh-seam coal is shown by Branin,[7] who writes:

> The largest single channel for river movement of coal is the Monongahela River . . . serving portions of the rich Klondike, Connellsville and Pittsburgh fields of western Pennsylvania and the Fairmont field of northern West Virginia, is the largest originator of coal tonnage. Total loadings in 1941 exceeded 25,000,000 tons, more than half of which was distributed along the river itself, including Pittsburgh. Originating and terminal tonnages for the Monongahela River are shown on the accompanying map (Fig. 1).

According to the Minerals Yearbook, total shipments by rail, river, truck and wagon from the five counties in Pennsylvania, where the Pittsburgh seam is the dominant producer, during 1941 were 68,675,347 tons. Thus over 35 per cent of this tonnage was loaded on the Monongahela River.

RESERVES OF THE PITTSBURGH SEAM

As of Jan. 1, 1937, the recoverable Pittsburgh-seam coal remaining in Pennsylvania, that tributary to river loading and that analyzing 1.5 per cent sulphur and less were estimated by Eavenson, Alford and Auchmuty (unpublished data) as shown in Tables 5, 6, and 7.

TABLE 5.—*Summary of Recoverable Pittsburgh-seam Coal Remaining in Pennsylvania January* 1, 1937

THOUSANDS OF NET TONS[a]

County	Total		Tributary to River Loading			All Other Tonnage	Truck Mines or Reservations,[b] Tonnage
	Tonnage	Distribution in Counties	Tonnage	Distribution in Counties	Percentage of County Total		
Allegheny	160,400	2.5	0	0.0	0.0	147,579	12,821
Fayette	315,096	4.8	190,610	11.9	60.5	119,599	4,887
Greene	2,983,192	45.8	1,012,288	63.1	33.9	1,970,702	202
Washington	2,798,055	43.0	401,299	25.0	14.3	2,391,319	5,437
Westmoreland	256,781	3.9	0	0.0	0.0	255,439	1,342
Total	6,513,524	100.0	1,604,197	100.0	24.6	4,884,638	24,689

[a] Data from Eavenson, Alford and Auchmuty, Mining Engineers, Pittsburgh, Pa.

[b] Reservations cover acreage reserved for surface support.

The tonnage listed as tributary to river loading was determined by owner control of access to a navigable river for acreage within an economic distance from such river for known transportation means other than common carrier railroads. Table 5 shows that a very considerable tonnage and a significant proportion (24.6 per cent of the total reserves) are tributary to river loading.

However, the reserves with a sulphur content of 1.50 per cent or less tributary to river loading are only about one half of this tonnage, and represent only 13.6 per cent of the total reserves. Fayette County has been the dominant producer of coking coal loaded on the river and most of this analyzed 1.20 per cent sulphur or less. From Table 7 it is clear that only about one fifth of the reserves of coal of 1.50 per cent sulphur or less that can be loaded on the river are in Fayette County. The remainder is about equally distributed between Washington and Greene Counties and has a higher average sulphur content than the Fayette County reserves.

Also, as indicated by note *b* on Tables 6 and 7, some coal higher than 1.50 per cent sulphur is included in the reserves listed as of 1.50 per cent sulphur and less, and in many of these areas selective mining must be done to maintain this limit of 1.50 per cent sulphur. Selective mining means the exclusion of about 6 in. of coal at the

TABLE 6.—*Summary of Recoverable Pittsburgh-seam Coal Remaining in Pennsylvania According to Sulphur Content, January* 1, 1937[a]

THOUSANDS OF NET TONS

County	Total without Regard to Sulphur Content		Analyzing 1.5 Per Cent Sulphur or Less[b]			
	Tonnage	Distribution in Counties	Tonnage	Distribution in Counties	Percentage of County Total	Percentage of Total Reserves
Allegheny	147,579	2.3	20,455	1.2	13.9	0.3
Fayette	310,209	4.8	310,209	18.7	100.0	4.8
Greene	2,982,990	46.0	502,809	30.4	16.9	7.7
Washington	2,792,618	43.0	627,638	37.9	22.5	9.7
Westmoreland	255,439	3.9	195,369	11.8	76.5	3.0
	6,488,835	100.0	1,656,480	100.0	25.5	25.5

[a] Reserves shippable by truck mines and reservations not included. Data from Eavenson, Alford and Auchmuty.

[b] The limiting factor in these reserves is generally a sulphur content before cleaning of 1.50 per cent or less, but some properties known to be furnishing coking coal have been included entirely, even though portions of the property are known to be higher in sulphur. In many of these areas selective mining must be done to maintain this sulphur content. Probably 60 per cent of the tonnage recorded is all that can be considered metallurgical coal, not over 1.25 per cent sulphur, without selective mining and mechanical cleaning.

TABLE 7.—*Summary of Recoverable Pittsburgh-seam Coal Remaining in Pennsylvania Analyzing* 1.5 *Per Cent Sulphur or Less and Tributary to River Loading, January* 1, 1937[a]

THOUSANDS OF NET TONS

County	Tonnage without Regard to Sulphur Content	Analyzing 1.5 % Sulphur or Less[b]				
		Total Tonnage	Tributary to River			
			Tonnage	Distribution in Counties	Percentage of County Total	Percentage of Total Reserves
Allegheny	147,579	20,455	0	0.0	0.0	0.0
Fayette	310,209	310,209	190,610	21.6	61.4	2.9
Greene	2,982,990	502,809	335,731	38.0	66.8	5.2
Washington	2,792,618	627,638	357,425	40.4	56.9	5.5
Westmoreland	255,439	195,369	0	0.0	0.0	0.0
Total	6,488,835	1,656,480	883,766	100.0	53.4	13.6

[a] Reserves shippable by truck mines and reservations not included. Data from Eavenson, Alford and Auchmuty.

[b] The limiting factor in these reserves is generally a sulphur content before cleaning of 1.50 per cent or less, but some properties known to be furnishing coking coal have been included entirely, even though portions of the property are known to be higher in sulphur. In many of these areas selective mining must be done to maintain this sulphur content. Probably 60 per cent of the tonnage recorded is all that can be considered metallurgical coal, not over 1.25 per cent sulphur, without selective mining and mechanical cleaning.

bottom of the seam and usually 8 in. or more of coal at the top from the mine production.

This outline of the economic importance of the Pittsburgh seam in Pennsylvania, and the indications of the diminishing reserves of coking coals of acceptable sulphur analysis tributary to cheap river transportation shows the necessity of studying the characteristics of the seam so that mining and beneficiation methods may be devised to make a larger proportion of these reserves available at a low cost to the needs of the industry of the country.

Characteristics of the Seam in Pennsylvania

The characteristics of the Pittsburgh seam are well known over a considerable extent of the seam in Pennsylvania. A large part of the seam in the Connellsville, Youghiogheny-Westmoreland, and Pittsburgh districts has been mined out, but private and published records provide sufficient data to outline areas of similar characteristics in respect to analyses, seam dimensions, structure and use. In the southwestern part of the state, particularly Greene County and the southern part of Washington County, the seam has been developed only near the Monongahela River. In some areas the seam was mined out before 1900 and data are meager, while in other localities, particularly near Pittsburgh, one company has made a complete study of the seam characteristics over an extensive area.

As a basis for discussion in respect to seam characteristics and analysis, the Pittsburgh seam in Pennsylvania may be classed into five general groups; each covering an area of wide extent: (1) Panhandle, (2) thin-vein Youghiogheny (Youghiogheny-Westmoreland districts), (3) thick-vein Youghiogheny (Youghiogheny-Westmoreland districts), (4) Connellsville, (5) Greene County (including the southern part of Washington County). There are several smaller areas, such as the Greensburg, Latrobe, Ligonier and Avella, whose individual characteristics have not been considered herein.

Within each general group are local areas where the characteristics are not in close agreement with the group average and the variations in the seam thickness and other characteristics may appear large to the reader. Mr. Eavenson[2] says:

> . . . but in view of the enormous area covered by this deposit and the tremendous quantity of it laid down originally, it is remarkable that its quality is so uniform. Many coal seams can show much greater consistency in some qualities than the Pittsburgh, but considering quality, thickness, and mining conditions together, few mineral deposits of any kind, anywhere, can rank with it.

In the Pittsburgh area the thickness and analytical characteristics of the Pittsburgh seam have been extensively studied by the Pittsburgh Coal Co. Some 31 mines in the area south and west of Pittsburgh and northwesterly from the Connellsville region were included in the study, and their location is shown by the map of Fig. 2. In this area complete analytical information was obtained for each mine sampled. From the data the mines are divided into three groups according to certain physical and chemical characteristics and were called Panhandle, Thin-Vein Youghiogheny, and Thick-Vein Youghiogheny.

Characteristics of Panhandle, Thin-vein and Thick-vein Youghiogheny Groups

Several hundred samples of each bench in 31 mines were collected. Samples were uniformly collected, prepared, and analyzed and therefore the results are directly comparable with one another in all respects. Each sample was collected from a particular horizontal section of the seam.

The standard method of sampling used consisted of making five benches of the seam: (1) the top 10 in. of the coal seam;

(2) the top breast, consisting of the coal between the top 10 in. and the top bearing-in band; (3) the bearing-in bands and coal between, which consisted of two slate partings, each approximately ½ to ¾ in. thick with 1½ to 4 in. of coal between the two slate bands; (4) the bottom breast, consisting of the coal between the bottom bearing-in band and the bottom 6 in.; (5) the bottom 6 in. of the coal seam. In calculating the analysis of the over-all seam the bearing-in bands and coal between were excluded, for reasons explained hereafter.

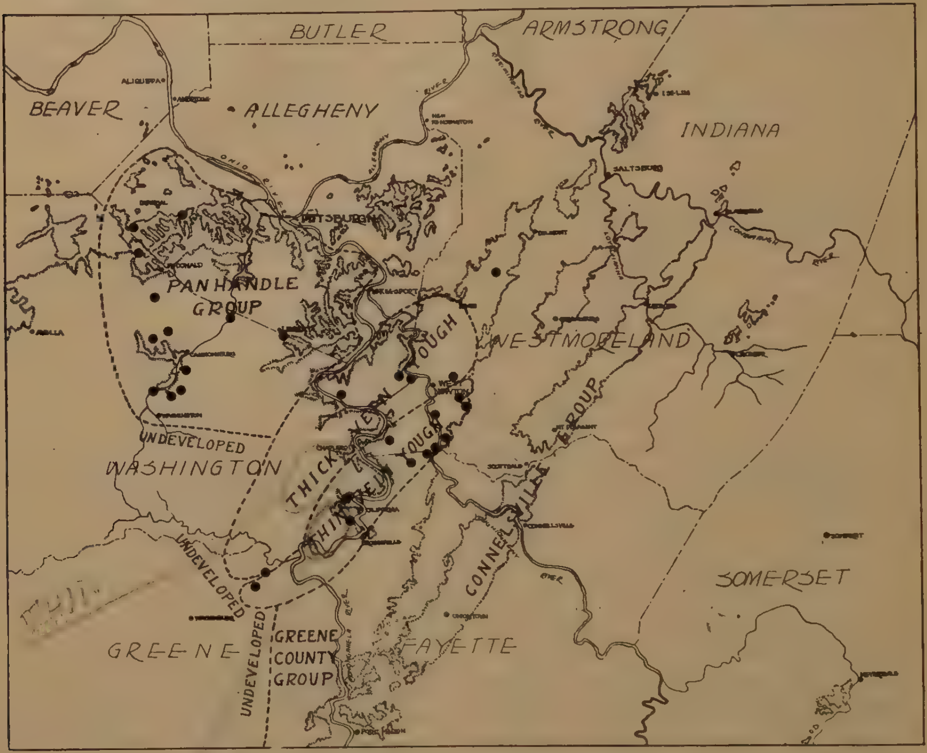


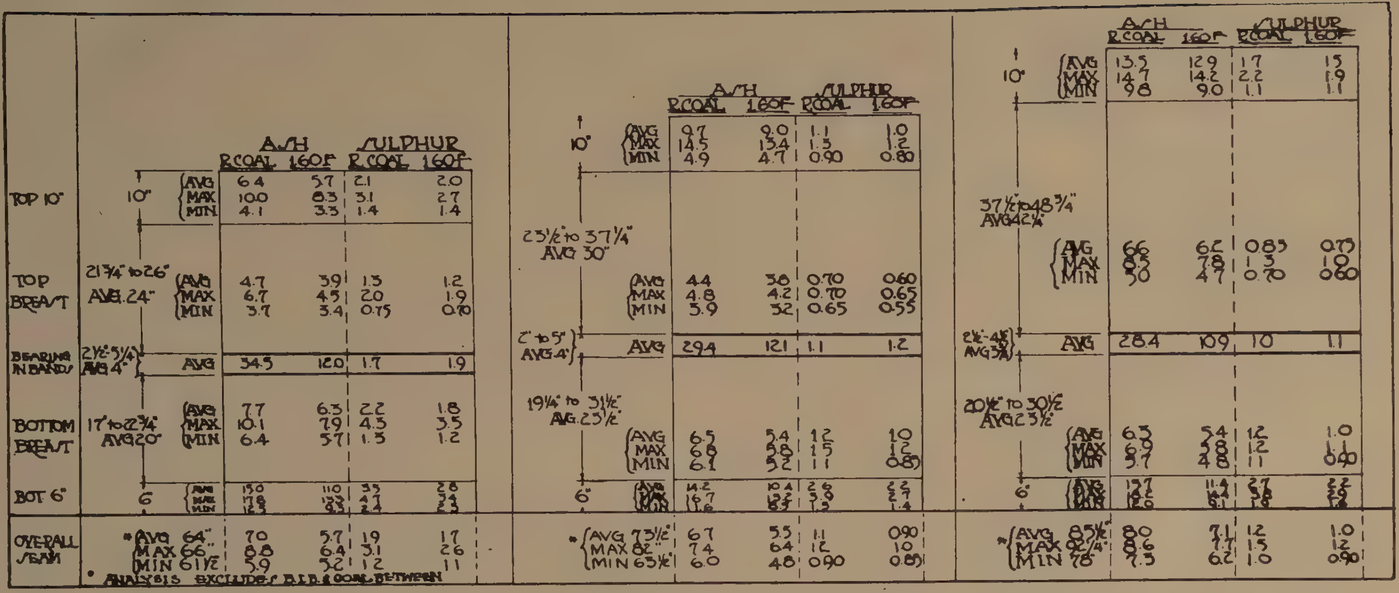
FIG. 2.—LOCATION OF MINES SAMPLED IN DETERMINING EXTENT OF PANHANDLE, THIN-VEIN YOUGH AND THICK-VEIN YOUGH GROUPS.
Black dots indicate mine openings.

THICK

Each sample was broken to pass a 4-in. round-hole screen and specific-gravity separation tests were run at 1.40 and 1.60 sp. gr. Only results at 1.60 sp. gr. have been reported herein. Because the samples were crushed to minus 4-in. before the separation tests were made, the results are closely comparable with results that would be obtained on a coal containing only the impurities present in the seam, which had received the size degradation of normal mining operations.

Fig. 3 shows the range of ash and sulphur by benches in the Panhandle, Thin-Vein and Thick-Vein Yough groups, while Table 8 shows a comparison of the average analysis of the over-all seam for the same three groups. From the analytical data derived from sampling and testing hundreds of samples, the following comparison of their characteristics has been prepared.

The thickness varies considerably. The Panhandle is the thinnest, averaging 64 in.;



a

			ASH R.COAL	ASH 1.60F	SULPHUR R.COAL	SULPHUR 1.60F
TOP 10"	10"	AVG	6.4	5.7	2.1	2.0
		MAX	10.0	8.3	3.1	2.7
		MIN	4.1	3.3	1.4	1.4
TOP BREAST	21¾" to 26" AVG. 24"	AVG	4.7	3.9	1.5	1.2
		MAX	6.7	4.5	2.0	1.9
		MIN	3.7	3.4	0.75	0.70
BEARING IN BANDS	2½"-5¼" AVG 4"	AVG	34.5	12.0	1.7	1.9
BOTTOM BREAST	17" to 22¾" AVG 20"	AVG	7.7	6.3	2.2	1.8
		MAX	10.1	7.9	4.5	3.5
		MIN	6.4	5.7	1.5	1.2
BOT 6"	6	AVG	15.0	11.0	3.3	2.8
		MAX	17.8	13.3	4.7	3.4
		MIN	12.3	8.3	2.4	2.3
OVERALL SEAM	*	AVG 64"	7.0	5.7	1.9	1.7
		MAX 66"	8.8	6.4	3.1	2.6
		MIN 61½"	5.9	5.2	1.2	1.1

* ANALYSIS EXCLUDES B.I.B. & COAL BETWEEN

b

			ASH R.COAL	ASH 1.60F	SULPHUR R.COAL	SULPHUR 1.60F
TOP 10"	10"	AVG	9.7	9.0	1.1	1.0
		MAX	14.5	13.4	1.5	1.2
		MIN	4.9	4.7	0.90	0.80
TOP BREAST	23½" to 37¼" AVG 30"	AVG	4.4	3.8	0.70	0.60
		MAX	4.8	4.2	0.70	0.65
		MIN	3.9	3.2	0.65	0.55
BEARING IN BANDS	2" to 5" AVG 4"	AVG	29.4	12.1	1.1	1.2
BOTTOM BREAST	19¼" to 31½" AVG. 23½"	AVG	6.5	5.4	1.2	1.0
		MAX	6.8	5.8	1.5	1.2
		MIN	6.1	5.2	1.1	0.85
BOT 6"	6"	AVG	14.2	10.4	2.6	2.2
		MAX	16.7	12.2	3.9	2.7
		MIN	11.6	8.5	1.3	1.4
OVERALL SEAM	*	AVG 73½"	6.7	5.5	1.1	0.90
		MAX 82"	7.4	6.4	1.2	1.0
		MIN 65½"	6.0	4.8	0.90	0.85

c

			ASH R.COAL	ASH 1.60F	SULPHUR R.COAL	SULPHUR 1.60F
TOP 10"	10"	AVG	13.5	12.9	1.7	1.5
		MAX	14.7	14.2	2.2	1.9
		MIN	9.8	9.0	1.1	1.1
TOP BREAST	37½" to 48¾" AVG 42¼"	AVG	6.6	6.2	0.85	0.75
		MAX	8.3	7.8	1.3	1.0
		MIN	5.0	4.7	0.70	0.60
BEARING IN BANDS	2¼"-4½" AVG 3¾"	AVG	28.4	10.9	1.0	1.1
BOTTOM BREAST	20½" to 30½" AVG 23½"	AVG	6.3	5.4	1.2	1.0
		MAX	6.9	5.8	1.2	1.1
		MIN	5.7	4.8	1.1	0.90
BOT 6"	6"	AVG	13.7	11.4	2.7	2.2
		MAX	14.2	14.4	3.8	2.9
		MIN	12.0	8.1	1.8	1.8
OVERALL SEAM	*	AVG 85½"	8.0	7.1	1.2	1.0
		MAX 92¼"	8.6	7.7	1.5	1.2
		MIN 78"	7.5	6.2	1.0	0.90

FIG. 3.—RANGE OF ASH AND SULPHUR IN THE PITTSBURGH SEAM (BY BENCHES), PANHANDLE, THIN-VEIN AND THICK-VEIN YOUGH GROUPS. *a*, Panhandle group, average 12 mines; *b*, Thin-Vein Yough, average 6 mines; *c*, Thick-Vein Yough, average 10 mines.

the Thin-Vein group is somewhat thicker, averaging 73½ in.; and the Thick-Vein Yough, averaging 85½ in., is the thickest. The most consistent bench is the bottom breast, which averages 23½ in. in thickness in both the Thin-Vein and Thick-Vein groups and 20 in. in the Panhandle.

The bearing-in bands have not been included in the analysis of the seam, as in hand-loading practice it has been customary to gob most of the bands underground. The sampling from which these data have been derived was started before mechanical loading had been adopted to any great extent and for that reason the bands were excluded from the seam analysis. The addition of the bands and coal between adds as much ash as the rest of the free impurities in the seam and their addition almost doubles the impurities (sink at 1.60 sp. gr.) of the coal. The effect of the removal of the bands on sulphur is negligible, usually increasing or decreasing the over-all seam sulphur by only a few hundredths of one per cent.

Fig. 3 shows that on an average there are only about 50 in. of low-ash coal, consisting of about 30 in. above and 20 in. below the bands. The rest of the coal is of intermediate quality. Studies have shown that if the top coal is considered to be the intermediate quality coal under the draw slate, the Panhandle group has practically no top coal, the Thin-Vein Yough has approximately 10 in., and the Thick-Vein Yough from 20 to 30 in. This is readily apparent in Fig. 4. The 30 in. of coal directly above the bands in the Thick-Vein is very similar in characteristics to the entire top breast (also 30 in.) of the Thin-Vein Yough group. In all three groups the part of the seam immediately above the bands is the lowest in ash and sulphur. The Greene County mine whose analysis also appears in Fig. 4 has higher ash and sulphur in the top breast then either of the three groups under discussion.

The uppermost 10 in. of the seam shows more variation in coal quality than any other part of the seam. The ash in the top 10 in. in the Panhandle group is low, being considerably less than that in the Yough groups, but when the top 10 in. of the Panhandle is compared with the portion of the seam that is at a corresponding distance above the bands in both Yough groups, the ash content is comparable in all three groups. In the Panhandle group, the mines in the northwestern part have the highest ash top coal in that group, due to two prominent slate bands occurring just under the draw slate. The top 10 in. of the Thick-Vein Yough group is high in ash, both in the raw coal and in the float at 1.60 sp. gr. In the Thin-Vein Yough group the coal in the top 10 in. is of intermediate quality but not as high in ash as the Thick-Vein group.

The bottom breast is the most consistent bench of the Pittsburgh seam for the three groups in the Pittsburgh district. On a pyrite-free basis, the bottom breast shows almost the same ash content for all three groups.

The bottom 6 in. is the worst part of the seam, ranging from 11.6 to 18.2 per cent ash and from 1.5 to 4.7 per cent sulphur (average for individual mines). Usually this coal is completely or partly left in the mine.

There is little difference in the average character of the bottom coal in the three groups, but it varies considerably between the individual mines. In the operation of a cleaning plant, the ash of the bottom 6 in. is slightly less objectionable than that in the top 10 in. of the Thick-Vein Yough group, but the sulphur is far more offensive, especially in the production of metallurgical coal. For all groups the sulphur in the bottom 6 in. is reduced approximately 20 per cent from the raw coal to float at 1.60 specific gravity.

The Panhandle group, as compared with the Thin-Vein and Thick-Vein groups (Table 8), is characterized by a thinner

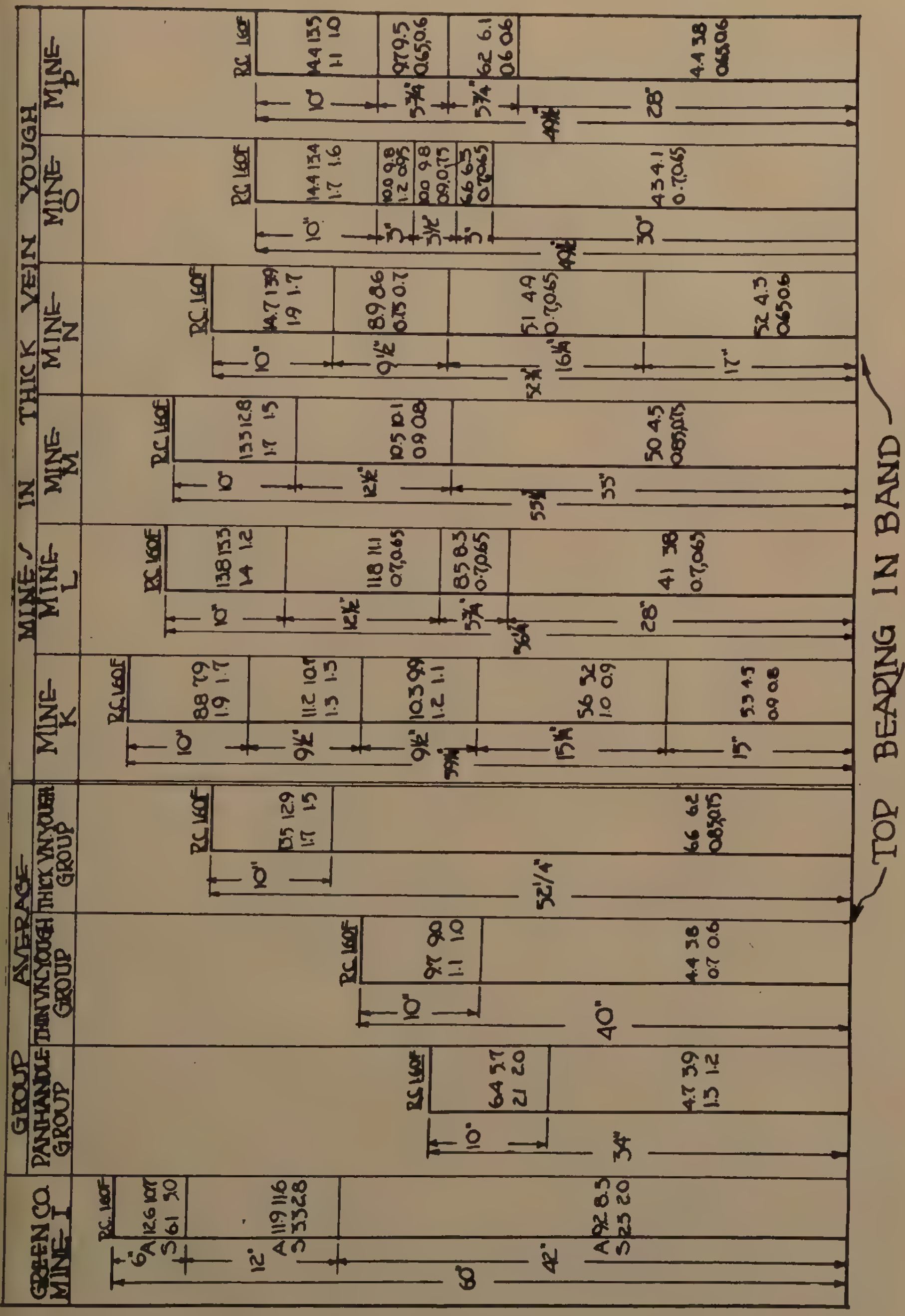


Fig. 4.—Variation in benches above bearing-in bands in Pittsburgh seam. *A*, per cent ash; *S*, per cent sulphur.

seam, an over-all seam ash comparable with the Thin-Vein Yough but lower than the Thick-Vein Yough, higher sulphur, slightly higher in volatile, and lower H-value (moisture-and-ash-free B.t.u.), a lower ash-softening temperature due to a higher iron content of the ash, and a coal of firmer structure. A greater percentage of the seam is taken out as block coal (plus 4-in.) in the mines of the Panhandle than in the mines of the Yough groups, because of the harder structure of the coal.

TABLE 8.—*Average Analysis of Over-all Seam for Panhandle, Thin-Vein Yough and Thick-Vein Yough Groups*

Group	Over-all Seam Sample (Excluding Bands and Coal Between), Raw Coal				
	Dry		Moisture-and-ash-free Basis		Ash-softening Temperature, Deg. F.
	Ash, Per Cent	Sulphur, Per Cent	Volatile, Per Cent	B.t.u.	
Panhandle..	7.0	1.9	41.2	15,090	2340
Thin-Vein Yough...	6.7	1.1	37.8	15,275	2470
Thick-Vein Yough...	8.0	1.2	37.3	15,335	2510

The Yough groups are more similar to one another than to the Panhandle group. Both groups are thicker than the Panhandle, the Thick-Vein Yough being of greater thickness (Figs. 3 and 4). Both Yough groups are low in sulphur, the Thin-Vein Yough containing 1.1 per cent in the over-all-seam raw coal, and the Thick-Vein Yough 1.2 per cent, as compared with 1.9 per cent sulphur average for the Panhandle. The over-all seam of the Thick-Vein Yough is higher in ash than either the Thin-Vein Yough or the Panhandle, the two latter being comparable in the ash of the over-all seam. The Thick-Vein Yough contains a higher ash and sulphur in the top coal and top breast than the Thin-Vein Yough. The two Yough groups are lower in volatile than the Panhandle, the Thick-Vein being slightly lower than the Thin-Vein (Table 8). Their H-values are considerably higher than those of the Panhandle, the Thick-Vein being somewhat higher than the Thin-Vein Yough. The ash-softening temperatures of the Yough groups are much higher than that of the Panhandle, the Thick-Vein being slightly higher than the Thin-Vein Yough. The Yough coals are generally known on the market as "gas" coals, the Thin-Vein coals going into the manufacture of gas and the Thick-Vein coals more generally used for by-product coke. The Panhandle coals have been used mostly as "steam" coals; the larger sizes for domestic fuel and gas manufacture. The firm structure of the Panhandle coals is responsible for a coarser product, and this has some effect on market preference.

The volatile content increases constantly toward the west and this is borne out by the progressive increase in volatile from the Thick-Vein to the Thin-Vein to the Panhandle (Table 8). The H-value progressively decreases in the same direction, the H-value decreasing 300 to 400 B.t.u. from the easternmost mine in the Thick-Vein group to the westernmost mine sample in the Panhandle. The structure of the coal also changes from east to west, the softest and most friable coal being in the Thick-Vein group and the firmest and hardest in the western edge of the Panhandle field.

These general statements refer only to the mines whose locations are indicated on Fig. 2, and whose characteristics were thoroughly studied.

GEOGRAPHICAL DISTRIBUTION OF SULPHUR

The geographical distribution of sulphur in the Pittsburgh seam of Pennsylvania is fairly well defined over a considerable extent of the seam. Because of the large number of sources used in collecting the information presented herein, the number of analyses and the uniformity of the methods of sampling and reporting of results vary considerably. The sulphur data are reported on a "dry" or "moisture-free" basis and are based on raw or unwashed run-of-mine coal "as mined." Where top coal (also called head coal) or bottom coal is unmined, the sulphur analyses refer only to the part of the seam removed. As the sources did not always disclose the proportion of the seam mined, the analyses are not necessarily on a common basis of seam section.

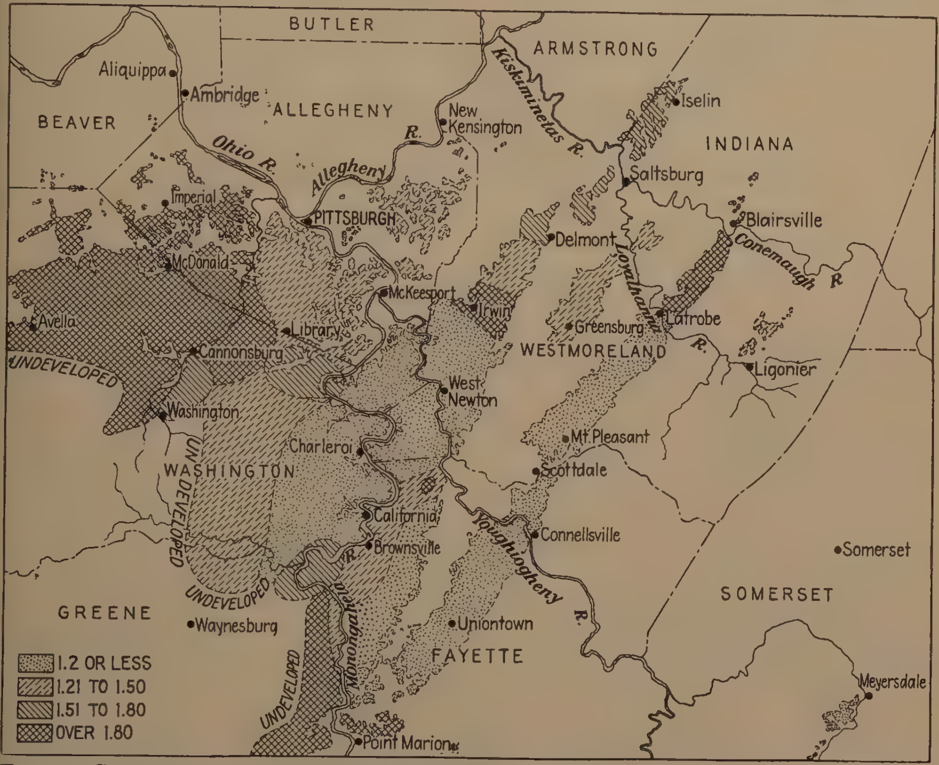


FIG. 5.—GEOGRAPHICAL DISTRIBUTION OF SULPHUR IN PITTSBURGH SEAM (COAL AS MINED) IN WESTERN PENNSYLVANIA.

From data available (refs. 2 and 8 to 12 as well as analyses made by the Pittsburgh Coal Co.), representing several hundred analyses, it has been possible to prepare a map showing the geographical distribution of sulphur (Fig. 5).

A large percentage of the coal containing 1.20 per cent sulphur or less has been mined out. Practically all of the coal under 1.2 per cent sulphur lies in Westmoreland and Fayette Counties. A relatively large area of coal containing 1.21 to 1.50 per cent sulphur is still being mined and developed for mining.

There is a rather small portion of the total Pittsburgh seam that contains from 1.51 to 1.80 per cent sulphur. The large reserves are over 1.80 per cent sulphur and in undeveloped acreage of unknown sulphur. The largest acreage of undeveloped Pittsburgh-seam coal is in Washington and Greene Counties, and from all indications and available information the raw coal is expected to contain more than

1.80 per cent sulphur, with possibly some from 1.51 to 1.80 per cent.

The locations of the boundary lines between areas of different average sulphur content in Fig. 5 are not exact enough so that individual properties or mines can always be definitely placed in one specific division or another. Variations within individual areas may occur, so that one mine property may not be represented by the average for its area. It is believed, however, that Fig. 5 is sufficiently accurate to indicate a close approximation of the geographical distribution of sulphur as it occurs in the Pittsburgh seam, each division indicating the range of sulphur within which the average for the area will fall.

FORMS OF SULPHUR

Sulphur is found in coal in three forms: (1) organic, (2) sulphate, and (3) pyritic.

TABLE 9.—*Distribution of Sulphur Forms in Specific-gravity Fractions of Various Sizes or Seam Samples*

Group, Mine, Size or Description	Sulphur,[a] Per Cent					
	1.30 Float	1.30 to 1.40	1.40 to 1.50	1.50 to 1.60	1.60 to 1.70	1.70 Sink
Pyritic and Sulphate Sulphur						
Panhandle, mine A, seam composite	0.34	1.35	2.05	3.22	2.76	8.55
Mine B, 1⅛ to ⅜-in.	0.60	2.09	4.48	5.99	7.29	12.09
Thin-Vein Yough, mine C, 5/16-in. to 20-mesh	0.27	0.67	0.91	0.95	1.12	1.54
Mine D, 5/16-in. to 20-mesh	0.33	0.65	0.95	1.12	1.19	1.99
Mine D, 1 to 5/16-in.	0.32	1.00	1.10	1.11	0.73	2.42
Mine E, 1⅛ to 5/16-in.	0.16	1.07	1.48	1.12	1.02	2.07
Thick-Vein Yough, mine F, 1⅛ to 5/16-in.	0.32	1.22	1.77	1.93	3.42	6.76
Mine G, seam composite	0.55	1.45	4.22	5.10	6.31[b]	8.97[c]
Greene County, mine H, 4-in. to 48-mesh	0.17	0.77	2.60	3.04	3.77	10.28
Mine I, 3-in. to 0, mine run	0.31	0.99	3.50	4.98	4.93	9.17
Organic Sulphur						
Panhandle, mine A, seam composite	0.79	1.06	0.86	0.83	0.69	0.56
Mine B, 1⅛ to ⅜-in.	1.38	1.64	1.87	1.55	1.18	0.13
Thin-Vein Yough, mine C, 5/16-in. to 20-mesh	0.65	0.88	0.80	0.58	0.43	0.21
Mine D, 5/16-in. to 20-mesh	0.80	0.84	0.82	0.75	0.46	0.00
Mine D, 1 to 5/16-in.	0.66	1.17	0.89	0.62	0.25	0.15
Mine E, 1⅛ to 5/16-in.	0.73	0.82	0.73	0.42	0.39	0.25
Thick-Vein Yough, mine F, 1⅛ to 5/16-in.	0.83	0.84	0.79	0.69	0.59	0.42
Mine G, seam composite	0.80	0.78	1.01	0.75	0.75[b]	0.64[c]
Greene County, mine H, 4-in. to 48-mesh	0.90	0.89	0.93	0.83	0.61	0.54
Mine I, 3-in. to 0, mine run	1.22	1.30	1.20	0.97	0.81	0.44
Total Sulphur						
Panhandle, mine A, seam composite	1.13	2.41	2.91	4.05	3.45	9.11
Mine B, 1⅛ to ⅜-in.	1.98	3.73	6.35	7.54	8.47	12.22
Thin-Vein Yough, mine C, 5/16-in. to 20-mesh	0.92	1.55	1.71	1.53	1.55	1.75
Mine D, 5/16-in. to 20-mesh	1.13	1.49	1.77	1.87	1.65	1.99
Mine D, 1 to 5/16-in.	0.98	2.17	1.99	1.73	0.98	2.57
Mine E, 1⅛ to 5/16-in.	0.89	1.89	2.21	1.54	1.41	2.32
Thick-Vein Yough, mine F, 1⅛ to 5/16-in.	1.15	2.06	2.56	2.62	4.01	7.18
Mine G, seam composite	1.35	2.23	5.23	5.85	7.06[b]	9.16[c]
Greene County, mine H, 4-in. to 48-mesh	1.07	1.66	3.53	3.87	4.38	10.82
Mine I, 3-in. to 0, mine run	1.53	2.29	4.70	5.95	5.74	9.61

[a] Moisture-free basis.
[b] 1.90 sp. gr. float.
[c] 1.90 sp. gr. sink.

The organic sulphur consists of organic compounds of sulphur almost uniformly distributed through the pure coal substance. In the Pittsburgh seam of Pennsylvania the known range of organic sulphur in the float at 1.30 sp. gr. is from 0.65 to 1.38 per cent (Table 9). For any one mine or group of several mines in the same general locality the organic sulphur usually falls within a rather narrow range. On the basis of available information, the Thin-Vein Yough group has the lowest percentage of organic sulphur, approximately 0.65 to 0.80 per cent, but in all probability that of the Connellsville region was as low if not lower than the Thin-Vein Yough. In both the Panhandle and Greene County groups, it is higher; the Panhandle group has a range of organic sulphur in the 1.30 sp. gr. float from approximately 0.79 to 1.38 per cent and Greene County from 0.90 to 1.22 per cent (Table 9). Some Greene County coal contains less than 0.90 per cent organic sulphur, but available data indicate that a considerable proportion of this coal will average 1.0 per cent or higher in organic sulphur.

TABLE 10.—*Comparison of Sulphur Forms in Specific-gravity Fractions between Thick-Vein Yough Mine F and Greene County Mine I, by Benches*

Description	Sulphur,[a] Per Cent											
	Pyritic			Sulphate			Organic			Total		
	1.30 Float	1.30 to 1.60	1.60 Sink	1.30 Float	1.30 to 1.60	1.60 Sink	1.30 Float	1.30 to 1.60	1.60 Sink	1.30 Float	1.30 to 1.60	1.60 Sink
Top Coal												
Thick-Vein Yough mine												
Section 1 (10 in.)	0.75	1.03	15.19	0.03	0.04	0.35	0.75	0.71	2.01	1.53	1.78	17.55
Section 2 (10-in.)	0.32	1.51	18.02	0.02	0.05	0.33	0.61	0.82	0.49	0.95	2.38	18.84
Greene County mine												
Section 3 (18-in.)	0.77	2.39	22.56	0.00	0.01	0.14	1.68	1.68	1.26	2.45	4.08	23.96
Section 4 (18-in.)	0.66	2.21	16.50	0.01	0.03	0.22	1.39	1.35	1.32	2.06	3.59	18.04
Breast Coal												
Thick-Vein Yough mine												
Section 1[b]	0.08	1.13	3.13	0.01	0.04	0.07	0.64	0.63	0.38	0.73	1.80	3.58
Section 2[b]	0.09	0.51	3.60	0.01	0.02	0.08	0.55	0.52	0.17	0.65	1.05	3.85
Greene County mine												
Section 3[c]	0.27	0.82	10.45	0.00	0.00	0.08	1.26	1.24	0.54	1.53	2.06	11.07
Section 4[c]	0.19	0.77	9.91	0.00	0.00	0.08	0.97	1.01	0.70	1.16	1.78	10.68
Bottom Coal												
Thick-Vein Yough mine												
Section 1	0.13	0.92	3.00	0.02	0.03	0.05	0.85	0.77	0.37	1.00	1.72	3.42
Section 2	0.19	1.29	9.84	0.02	0.04	0.21	0.74	0.85	0.28	0.95	2.18	10.33
Greene County mine												
Section 3	0.30	3.33	10.86	0.00	0.02	0.09	1.28	1.30	0.71	1.58	4.65	11.66
Section 4	0.40	1.66	14.66	0.00	0.00	0.09	1.20	1.23	1.06	1.60	2.89	15.81

[a] Moisture-free basis.
[b] Excludes bearing-in bands.
[c] Includes bearing-in bands.

When considering each mine individually, the difference in organic sulphur between the same specific gravity fractions of different benches is usually not of great magnitude, but both Table 10 and Table 11 show that benches of top coal and bottom coal are somewhat higher in organic sulphur than the breast coal of the same mine. The range of organic sulphur within the bottom breast of the Thick-Vein Yough mine (*F*) was 0.68 to 0.71 per cent and within the top coal it was 0.76 to 1.08 per cent (Table 11).

For most mines studied, the organic sulphur is not significantly lower in the float at 1.30 sp. gr. than in the 1.30 to 1.60 sp. gr. fraction, indicating that a coal of lower organic sulphur could not generally be obtained by cleaning at a low gravity.

Even where a reduction in organic sulphur could be obtained, a separation at 1.30 sp. gr. is impractical from an economic standpoint. Since coal cleaning of the Pittsburgh seam usually is carried out at or above 1.50 sp. gr., and because the fractions heavier than that decline in organic sulphur, the organic sulphur of the cleaned coal will be higher than that of the raw coal from which it is obtained.

TABLE 11.—*Average[a] and Range of Sulphur Forms[b] in Pittsburgh Seam by Benches*
THICK-VEIN YOUGHIOGHENY MINE F AND GREENE COUNTY MINE I

	Pyritic, Per Cent		Sulphate, Per Cent		Organic, Per Cent		Total, Per Cent	
	Thick-Vein Yough Mine F	Greene Co. Mine I	Thick-Vein Yough Mine F	Greene Co. Mine I	Thick-Vein Yough Mine F	Greene Co. Mine I	Thick-Vein Yough Mine F	Greene Co. Mine I
Top Coal[c]								
Average	1.85	2.75	0.03	0.02	0.85	1.45	2.73	4.21
Maximum	4.50	3.21	0.08	0.04	1.08	1.61	5.66	4.56
Minimum	0.64	2.35	0.01	0.00	0.76	1.31	1.49	3.76
Top Breast								
Average	0.49	1.09	0.01	0.01	0.66	1.23	1.16	2.33
Maximum	1.20	1.27	0.02	0.01	0.69	1.34	1.91	2.43
Minimum	0.11	0.96	0.00	0.00	0.63	1.09	0.75	2.21
Bottom Breast[d]								
Average	0.32	0.81	0.00	0.00	0.69	1.03	1.01	1.84
Maximum	0.64	1.07	0.01	0.01	0.71	1.11	1.34	2.18
Minimum	0.16	0.46	0.00	0.00	0.68	0.89	0.85	1.35
Bottom 6 Inches								
Average	0.95	3.48	0.02	0.02	0.88	1.22	1.85	4.72
Maximum	1.14	5.13	0.03	0.03	1.01	1.30	2.17	6.32
Minimum	0.74	2.13	0.01	0.01	0.81	1.17	1.56	3.38
Seam								
Average	0.69	1.41	0.01	0.01	0.72	1.19	1.42	2.61
Maximum	1.34	1.58	0.02	0.01	0.79	1.27	2.15	2.34
Minimum	0.69	1.21	0.00	0.00	0.70	1.12	1.06	2.82

[a] Mine F was represented by five samples; mine I by four samples.
[b] Moisture-free basis.
[c] Top coal of mine F was 10 in. and of mine I, 18 in. Other benches of approximately equal thickness.
[d] Mine F: bearing-in bands not included in bottom breast. Mine I: bearing-in bands included in bottom breast.

The sulphate form is a small percentage of the total sulphur and in most coal substances is negligible, usually amounting to 1 to 2 per cent of the total sulphur present and varying from 0.00 to 0.05 per cent in the 1.30 sp. gr. float and 1.30 to 1.60 sp. gr. fractions (Table 10). As a rule freshly mined coal contains little sulphate sulphur, but on exposure to the weather it increases because of oxidation of pyrite. Sulphate sulphur ranges usually from 0.05 to 0.35 per cent in the 1.60 sp. gr. sink fraction of coals from the Pittsburgh seam.

Pyritic sulphur occurs as relatively coarse pyrites and also as microscopic pyrites,[13] the latter being intimately mixed with the coal substance. The distribution of pyritic sulphur in respect to particle size will determine for each coal the percentage of pyritic sulphur that can be removed by coal-cleaning methods. The amount of pyritic sulphur is much more variable than that of organic or sulphate sulphur; it ranges from 0.10 per cent to as high as 1.60 per cent in the float at 1.30 sp. gr.; 0.50 per cent to 3.5 per cent in the 1.30 to 1.60 sp. gr. fraction, and 1.0 to 20.0 per cent in the sink at 1.60 specific gravity.

In the Pittsburgh seam all three forms are more highly concentrated in the top and bottom coals than in the breast coals and this is especially true for pyritic sulphur. A marked concentration of pyritic sulphur occurs in the top coal and bottom 6 in. for both the Thick-Vein Yough mine *F*, and Greene County mine *I* (Table 11). In the top coal of Greene County mine *I*, the pyritic sulphur averages 2.75 per cent with a range from 2.35 to 3.21 per cent on four samples. In the bottom 6 in. it averaged 3.48 per cent with a range from 2.13 to 5.13 per cent on four samples. Generally there is a tendency toward an increase in the percentage of pyritic and sulphate in the finer sizes as compared with the larger sizes.

From the available data, the approximate percentage of various forms of sulphur in raw coal of the Pittsburgh seam in Pennsylvania is summarized in Table 12.

TABLE 12.—*Range of Sulphur Forms in Raw Coal from the Pittsburgh Seam of Pennsylvania*
PANHANDLE, THIN-VEIN YOUGH, THICK-VEIN YOUGH, AND GREENE COUNTY GROUPS

Group	Sulphur[a] in Raw Coal, Per Cent			
	Pyritic	Sulphate	Organic	Total
Panhandle	0.25 to 2.20	0.01 to 0.05	0.90 to 1.40	1.20 to 3.20
Thin-Vein Yough	0.15 to 0.50	0.01 to 0.02	0.65 to 0.80	0.90 to 1.20
Thick-Vein Yough	0.20 to 0.80	0.01 to 0.02	0.75 to 0.85	1.00 to 1.50
Greene County	0.20 to 1.60	0.01 to 0.05	0.80 to 1.25	1.20 to 2.80

[a] Moisture-free basis.

PHOSPHORUS IN THE PITTSBURGH COAL SEAM

Phosphorus in the Pittsburgh coal seam varies widely both vertically from bench to bench and horizontally over comparatively small areas. The Panhandle coals are generally 0.015 per cent phosphorus or below while in the Thin-Vein and Thick-Vein Yough groups phosphorus varies from 0.010 to 0.025 per cent, with a greater variation from mine to mine than in the Panhandle. For the Connellsville group and the part of the Greene County group that is being mined, the amount of phosphorus is generally medium (0.010 to 0.015 per cent) or low (under 0.010 per cent).

The phosphorus of the over-all seam (excluding bands and coal between) for eight mines in the Thin-Vein Yough and Thick-Vein Yough groups varies from 0.010 to 0.025 per cent (Table 13). Phosphorus usually is highest in the top 10 in. and bottom 6 in. and often it will be very highly concentrated in either or both of these benches. The bottom breast is consistently the lowest in phosphorus content for all mines of the Yough groups shown in Table 13. The top breast, top 10 in., and bottom 6 in. show wide variations in phosphorus from mine to mine.

A more extensive breakdown of the seam of two high-phosphorus mines (Nos. 4 and 5 in Table 13) shows that the 30 in. above the bands and the 25 in. below the bands are similar in phosphorus content to corresponding benches of the other mines. Mine No. 4 shows as high as 0.130 per cent phosphorus in a bench 28 to 34 in. above the bands. Where the top breast is 28 in. or less in height in these two mines the phosphorus of the top breast is consistently lower than when its thickness is over 28 in. On the other hand, the less the thickness of the top breast, the greater the phosphorus content of the top 10 in. The phosphorus of the over-all seam for these two mines, however, is of the same magnitude.

The wide variation of phosphorus horizontally within a small area is illustrated by Table 14, showing bench analyses of sections A and B, which are different locations in the same mine of the Thin-Vein Yough group. The over-all seam analyses are 0.016 and 0.039 per cent phosphorus, respectively, for sections A and B raw coal. The wide variations in the phosphorus content of corresponding benches is to be noted in Table 14.

TABLE 13.—*Distribution of Phosphorus in the Pittsburgh Seam, by Benches*
THIN-VEIN AND THICK-VEIN YOUGHIOGHENY GROUPS

Description and Specific Gravity	Thin-Vein Yough Group				Thick-Vein Yough Group			
	Mine No.							
	1	2	3	4	5	6	7	8
Top 10 Inches								
Raw coal	0.061	0.032	0.015	0.075	0.026	0.027	0.017	0.006
1.60 float	0.062	0.032	0.013	0.071	0.026	0.026	0.015	0.005
1.60 sink	0.043	0.048	0.024	0.086	0.033	0.050	0.029	0.015
Top Breast								
Raw coal	0.004	0.007	0.018	0.013	0.029	0.023	0.009	0.015
1.60 float	0.003	0.006	0.018	0.011	0.027	0.023	0.008	0 015
1.60 sink	0.025	0.028	0.041	0.026	0.090	0.045	0.030	0.018
Bands and Coal Between								
Raw coal	0.008	0.014	0.012	0.012		0.079	0.016	0.004
1.60 float	0.006	0.009	0.009	0.007		0.006	0.007	0.004
1.60 sink	0.009	0.026	0.028	0.022		0.014	0.038	0.004
Bottom Breast								
Raw coal	0.003	0.007	0.008	0.004	0.005	0.007	0.005	0.002
1.60 float	0.003	0.007	0.007	0.003	0.003	0.006	0.004	0.002
1.60 sink	0.012	0.029	0.019	0.029	0.022	0.027	0.027	0.010
Bottom 6 Inches								
Raw coal	0.030	0.029	0.032	0.071	0.078	0.034	0.028	0.045
1.60 float	0.020	0.017	0.019	0.029	0.025	0.017	0.013	0.023
1.60 sink	0.147	0.136	0.169	0.298	0.284	0.181	0.058	0.203
Seam Excluding Bands								
Raw coal	0.016	0.014	0.016	0.023	0.025	0.020	0.010	0.013
1.60 float	0.014	0.011	0.014	0.017	0.020	0.018	0.008	0.012
1.60 sink	0.059	0.042	0.042	0.149	0.147	0.051	0.031	0.096

A study of the washability of phosphorus in the over-all seam of the Thin-Vein and Thick-Vein Yough groups (Table 13) indicates an average reduction of 16.5 per cent from raw coal to float at 1.60 sp. gr. The maximum decrease in phosphorus was 26.1 per cent on a high-phosphorus coal (mine No. 4) and the minimum (7.7 per cent) was made on a coal of medium phosphorus content (mine No. 8). The phosphorus in the top 10 in. and middle benches may be reduced slightly at 1.60 sp. gr., usually 0.001 to 0.002 per cent phosphorus. The bottom 6 in. offers the greatest opportunity for reduction of phosphorus by coal cleaning and a 40 to 70 per cent reduction from raw coal to float at 1.60 sp. gr. is indicated. The sink at 1.60 sp. gr. of the bottom 6 in. is usually high, ranging from 0.058 to 0.269 per cent phosphorus. In some places, the fire clay immediately underlying the seam has been found to contain as high as 0.510 per cent phosphorus. In many mines the bottom 4 to 6 in. is left in the mine because of its high ash and sulphur content, so that with this coal excluded the reduction in phosphorus of the entire seam by coal cleaning would be considerably less. If the entire bottom 6 in. were left in the mine, the percentage reduction of phosphorus by coal cleaning in the Thin-Vein Yough and Thick-Vein Yough groups would be slight.

TABLE 14.—*Variation in Phosphorus, by Benches, in Seam*

SECTIONS FROM A THIN-VEIN YOUGH MINE

Bench	Phosphorous, Per Cent			
	Section A		Section B	
	Raw Coal	1.60 Sp. Gr. Float	Raw Coal	1.60 Sp. Gr. Float
Top 10 in.	0.030	0.030	0.190	0.194
Next 10 in.	0.056	0.051	[a]	[a]
Next 28 in.	0.004	0.003	0.009	0.007
Next 25 in.	0.003	0.003	0.005	0.003
Next 2 in.	0.004	0.003	0.011	0.007
Bottom 6 in.	0.035	0.018	0.080	0.016
Over-all seam.	0.016	0.013	0.039	0.033

[a] This "bench" is missing, since seam is lower in section B.

TABLE 15.—*Distribution of Phosphorus by Screen Sizes for a Thin-Vein Yough Mine and an Average of 14 Mines of the Pittsburgh Seam in Pennsylvania*

PER CENT

Round-hole Screen Opening, In.	A Thick-Vein Yough Mine	Average of 14 Mines, Pittsburgh Seam in Pennsylvania
+6	0.080	0.029
6–4	0.075	0.020
4–2	0.039	0.015
2–1⅛	0.025	0.011
1⅛–⅜	0.021	0.009
Through ⅜	0.022	0.010

The distribution of phosphorus into various sizes will depend to a considerable extent upon the mining practices, especially in the manner in which the coal is cut or sheared and the method of breaking it down before loading. Those are the factors that determine the size of the coal produced from each of the various benches, and, since phosphorus varies widely in the different benches, they determine the sizes into which the phosphorus will be distributed. At one mine in the Thick-Vein Yough group the distribution of phosphorus is as indicated by Table 15. The coal was undercut and the larger sizes were produced from the upper part of the seam, thus the sizes over 2-in. round hole, and particularly over 4 in., were much the higher in phosphorus. By overshooting the bottom breast and screening at 2 in., a lower phosphorus content in the minus 2-in. coal was produced for metallurgical use and the plus 2 in. of higher phosphorus was used for the manufacture of gas. The average of 14 mines of the Pittsburgh seam of Pennsylvania generally indicates a higher phosphorus in the larger sizes. In most of these mines the coal was bottom-cut with no shearing.

GREENE COUNTY COAL (INCLUDING SOUTHERN WASHINGTON COUNTY)

Data on sulphur and other analytical properties of Greene County coal have been limited because of the generally heavy cover over the coal seam, the absence of outcrops except near the Monongahela River, and chiefly because of the small proportion of the seam developed. Until a few years ago the only data consisted of analyses of drill cores scattered over wide areas. These drill cores generally showed higher sulphur than coal from Fayette County, although low and medium-sulphur coal is being mined in the northeast part of the county. Mines in northern West Virginia and in the Point Marion district of Pennsylvania are generally high in sulphur, so that there has been little cause for believing that Greene

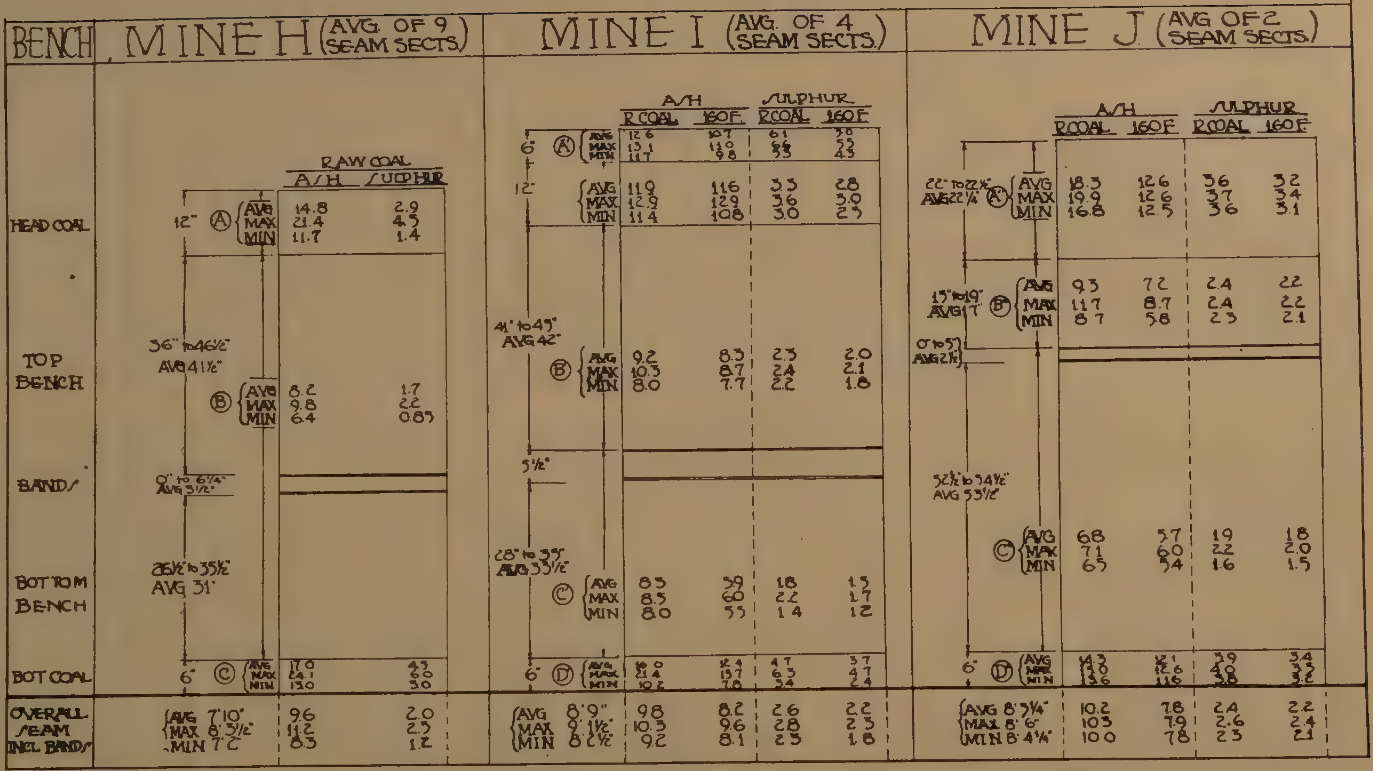

BENCH
MINE H (AVG. OF 9 SEAM SECTS.)
MINE I (AVG. OF 4 SEAM SECTS.)
MINE J (AVG. OF 2 SEAM SECTS.)
HEAD COAL
TOP BENCH
BANDS
BOTTOM BENCH
BOT COAL
OVERALL SEAM INCL. BANDS
RAW COAL
ASH SULPHUR
12" Ⓐ AVG MAX MIN 14.8 21.4 11.7 2.9 4.5 1.4
36" to 46½" AVG 41½"
Ⓑ AVG MAX MIN 8.2 9.8 6.4 1.7 2.2 0.85
0" to 6¼" AVG 3½"
26½" to 35½" AVG 31"
6" Ⓒ AVG MAX MIN 17.0 24.1 13.0 4.5 6.0 3.0
AVG 7'10" MAX 8'3½" MIN 7'2" 9.6 11.2 8.3 2.0 2.5 1.2
ASH SULPHUR
R.COAL 160F. R.COAL 160F.
6" Ⓐ AVG MAX MIN 12.6 13.1 11.7 10.7 11.0 9.8 6.1 6.6 5.5 5.0 5.5 4.5
12" AVG MAX MIN 11.9 12.9 11.4 11.6 12.9 10.8 3.3 3.6 3.0 2.8 3.0 2.5
41" to 45" AVG 42"
Ⓑ AVG MAX MIN 9.2 10.5 8.0 8.5 8.7 7.7 2.5 2.4 2.2 2.0 2.1 1.8
5½"
28" to 35" AVG 33½"
Ⓒ AVG MAX MIN 8.5 8.5 8.0 5.9 6.0 5.5 1.8 2.2 1.4 1.5 1.7 1.2
6" Ⓓ AVG MAX MIN 18.0 21.4 10.2 12.4 13.7 7.8 4.7 6.3 3.4 3.7 4.7 2.4
AVG 8'9" MAX 9'1½" MIN 8'2½" 9.8 10.5 9.2 8.2 9.6 8.1 2.6 2.8 2.5 2.2 2.5 1.8
ASH SULPHUR
R.COAL 160F. R.COAL 160F.
22" to 22½" AVG 22¼" Ⓐ AVG MAX MIN 18.5 19.9 16.8 12.6 12.6 12.5 3.6 3.7 3.6 3.2 3.4 3.1
15" to 19" AVG 17" Ⓑ AVG MAX MIN 9.3 11.7 8.7 7.2 8.7 5.8 2.4 2.4 2.3 2.2 2.2 2.1
0" to 5" AVG 2½"
52½" to 54½" AVG 53½"
Ⓒ AVG MAX MIN 6.8 7.1 6.5 5.7 6.0 5.4 1.9 2.2 1.6 1.8 2.0 1.5
6" Ⓓ AVG MAX MIN 14.3 13.0 13.6 12.1 12.6 11.6 3.9 4.0 3.8 3.4 3.2 3.2
AVG 8'5¼" MAX 8'6" MIN 8'4¼" 10.2 10.3 10.0 7.8 7.9 7.8 2.4 2.6 2.3 2.2 2.4 2.1

Fig. 6.—Range of ash and sulphur in the Pittsburgh seam of Greene County.

County coal as a whole would be of low or medium sulphur. In recent years additional information has been obtained on this coal, and such information as is available to the authors will be presented herein. Seam sections from three mines are included but it is not to be inferred that these are representative of the entire county.

The seam is considerably thicker than the Panhandle, Thin-Vein Yough, or Thick-Vein Yough groups and is comparable to the Connellsville coal in this respect (Fig. 6). The bearing-in bands are higher in the seam and are not as regular in respect to their location within the seam, nor are the bands as prominent or as thick as in the Panhandle, Thin-Vein Yough, and Thick-Vein Yough groups.

The top coal, or head coal, as it is generally called in this district, is high in ash and sulphur both in the raw coal and float at 1.60 sp. gr. (Fig. 6). In most mines the practice is to leave the head coal to support the draw slate. The top and bottom breasts are much higher both in ash and sulphur than the top and bottom breasts of the Panhandle, Thin-Vein and Thick-Vein Yough groups. The seam sections of Fig. 6 show higher sulphur than the average so far obtained in the part of Greene County extensively mined along the Monongahela River. Recent examinations by the U. S. Bureau of Mines[14] indicate that much of the pyritic sulphur in specimens of Greene County coal is too finely divided for removal by ordinary mechanical cleaning.

EFFECT OF SELECTIVE MINING AND MINING PRACTICES ON QUALITIES OF COAL PRODUCED

Because of the depletion of the better quality coal of the Pittsburgh seam, coal of higher ash and sulphur will be increasingly depended upon for the production of coke for the blast furnace. Because of its access to river transportation, Greene County undoubtedly will be a large producer of metallurgical coal. Since all available information indicates that the Panhandle and Greene County coals are higher in both organic and pyritic sulphur than coals formerly used from the Connellsville and Youghiogheny districts, new problems in sulphur reduction will be presented. The problem will require careful study for each mine and will involve the study of mining methods, face-preparation practices, and selective mining in their relation to the improvements in analytical properties that may be effected by coal cleaning. Only by coordinating the underground practices and the coal-cleaning program will it be possible to obtain the maximum economical reduction in ash and sulphur.

The following factors will each have its effect on the analysis of the coal produced:

1. Selective mining in respect to areas of various levels of quality.
2. Selective mining in respect to the proportion of the seam mined.
3. The method of loading employed.
4. The method of cutting and/or shearing.
5. The method of breaking down the coal.
6. The amount of refuse loaded with the coal.

Selective mining may consist of mining certain areas of higher quality coal in the first years of the mine's production, but this inevitably leads to an exhaustion of the better quality coal, leaving the coal of higher ash and sulphur for the remaining life of the mine. Some mines have operated without mechanical cleaning by following such a system of selective mining in the first years of operation and at some later date have installed mechanical cleaning equipment for beneficiation of the remaining coal of lower quality.

Another type of selective mining is the practice of mining only certain benches of the seam, leaving benches of inferior coal in the mine. The practice of most companies in the Panhandle, Thin-Vein Yough and Thick-Vein Yough areas has been to leave the bottom 4 to 6 in. of the seam in the mine, thereby producing a product of lower ash and sulphur than if the entire seam were removed. In the Connellsville and Greene County district the coal is of sufficient height to permit from 10 to 12 in. of head coal immediately below the draw slate to be left in place and thereby provide better roof conditions during mining. Since this bench is higher in ash and sulphur than the top and bottom breasts, the raw coal produced is lower in ash and sulphur than if the head coal had been mined. In addition, leaving the head coal in place has reduced the contamination of coal by the draw slate that overlies the seam. This practice is not followed in the Panhandle and Thin-Vein Yough groups because of the low height of the seam; in the Thick-Vein Yough group the top coal is held up in some mines.

The use of mechanical loading equipment is bringing new problems that never existed with hand loading. In some cases it has been found impossible to mine selectively in respect to certain benches with mechanical loading, as had been the practice with hand loading. This has been particularly true of the bottom coal, and its inclusion in the coal increases the ash and sulphur appreciably. With top cutting, the blasting of the coal usually carries all the way to the fire-clay bottom and the inferior bottom 6 in. is usually broken up and loaded with coal from the other benches. If the bottoms are soft or wet and therefore very friable, it is often difficult to avoid their inclusion with the loaded coal even when the undercut is held 6 in. above the bottom of the seam. In some places, under the hand-loading system, where the close timbering permitted head coal to be left in place to support the draw slate, it has been found impossible to adopt mechanical loading without taking down both head coal and draw slate. This often means that part or all of the head coal and draw slate is loaded out with the coal. As head coal is of intermediate quality, with inherently higher ash and sulphur, mechanical cleaning can only partially improve it. Thus by a change in the mining system or in methods of operation, the characteristics of the coal in respect to sizing, specific gravity, and analyses may be entirely changed.

Not only are the average characteristics often changed by the use of mechanical loading equipment, but the uniformity in respect to ash and sulphur may be adversely affected. With hand loading the production originates from a large number of working places and there is a thorough mixing of all coal originating from the one section. With mechanical loading an entire working place of from 20 to 40 or more tons is loaded out in a short time, with the result that coal from a single place is concentrated in a trip of cars. This may or may not be serious, depending upon the variability of the coal from working place to working place. Some mines, particularly those of low sulphur, show little variation from place to place within a section and the influence of concentrating the production from one working place in one trip of cars does not have much effect on the uniformity. In many of the higher sulphur mines the variation in sulphur from place to place in the seam shows a wide range, and mechanical loading has had an adverse effect on the uniformity of the ash and sulphur in the raw coal.

It has been shown previously that from one bench to another at the same location the variation in ash, sulphur and other analytical properties may be fairly large. When a prominent bone or slate band splits the better quality coal into two benches, it has sometimes been found advantageous to cut out this band and load it separately, so that it will not become mixed with the better quality coal of the other benches. A certain bench may require a greater amount of crushing than others to permit mechanical cleaning and it may be found expedient to cut in this bench. Certain sizes of coal may be produced from a given bench in the seam and the analytical properties of this bench may be suited for a particular purpose, while another bench may have properties more desirable for another use. By coordination of the cutting and shooting with the outside screening facilities, it may be possible to prepare coal more suitable to certain requirements than if the underground and outside preparation were entirely unrelated.

The amount of refuse loaded with the coal has a decided influence on the results obtained by the coal-cleaning equipment and particularly in the fine coal under ⅜ in. in size. Although the technical problems of cleaning fine coal down to even 100 or 150 mesh have been solved, the economics of coal cleaning must also be considered in relation to the removal of refuse from various sizes. The sizes below ⅜ in. require a greater investment in coal-cleaning equipment per ton of capacity than do coal sizes over ⅜ in. The contamination of the water system of the cleaning plant with high-ash slurry will result in either higher ash cleaned coal (unless froth flotation is employed) or in a higher refuse loss if these solids are wasted. Therefore in order to minimize the problems and expense of cleaning the sizes under ⅜ in. and to produce a cleaned coal of the highest quality with economy, it will be found advantageous to prevent undue contamination of the fines in the mining operation. It cannot be too strongly emphasized that the matter of realizing a given analytical specification for any particular coal depends to a considerable extent on the coordination between the mechanical cleaning program and the underground mining system and practice.

TABLE 16.—*Variation of Ash in Minus 4-inch Raw Coal before and after Passing through Storage Bins*

THIN-VEIN YOUGH MINE—MECHANICAL LOADING

Ash, Moisture-free Basis, Per Cent							
Before the Bins				After the Bins			
Average	Maximum	Minimum	Range	Average	Maximum	Minimum	Range
9.5	20.5	6.6	13.9	9.5	11.9	7.3	4.6

Deviation from Average, Per Cent	Percentage of Results	
	Before the Bins	After the Bins
± 0.5	13.7	40.2
± 1.0	33.3	70.6
± 2.0	76.5	98.0
± 3.0	89.2	99.0
± 5.0	96.0	100.0
±10.0	99.0	
±15.0	100.0	

TABLE 17.—*Improvement in Uniformity of Cleaned Coal Produced by Controlled Dumping of Mine Cars at a Greene County Mine*

Raw Coal to Cleaning Plant	Total Sulphur,[a] Per Cent, in Cleaned Coal			
	Average[b]	Maximum	Minimum	Range
Mixed through mixing bins	1.33	1.68	1.13	0.55
Mixed through mixing bins plus controlled dumping of mine cars	1.35	1.41	1.17	0.24

[a] Moisture-free basis.

[b] Coal produced from the same sections during successive periods of operation.

EFFECT ON UNIFORMITY OF MIXING AND CONTROLLING OUTPUT BY ANALYSES

In a uniform coal the analytical values are distributed over a smaller range than in a highly variable coal. To express variability, or, conversely, uniformity, the usual practice is to show the maximum variation between high and low values. This is often called the "range." Another method is to tabulate the percentage of samples that falls within certain limits of the mean of a number of samples analyzed.

TABLE 18.—*Mechanical Cleaning Plants Treating Pittsburgh-seam Coal in Pennsylvania*

Year Started	Capacity, Tons per Hour	Size Cleaned, In.	Kind of Cleaning Unit	Mine or Coke Plant	Company
1913	150	¾–0	Foust jigs	Aliquippa[a]	Jones and Laughlin Steel Corp.
1914	50	¾–0	Foust jigs	Aliquippa[a]	Jones and Laughlin Steel Corp.
1917	50	¾–0	Foust jigs	Aliquippa[a]	Jones and Laughlin Steel Corp.
1923	150	¾–0	Foust jigs and Overstrom wet tables	Salem No. 1	Keystone Coal and Coke Co.
1927	50	¾–0	Pittsburgh jig	Latrobe[b]	Saxman Coal and Coke Co.
1927	300	2 –0	Arms air tables	Champion No. 3	Pittsburgh Coal Co.
1928	200	4 –0	Menzies hydros and Arms air tables	Clinton block	Clinton Block Coal Co.
1928	250	4 –0	Peale-Davis air tables	Lincoln No. 1[c]	Lincoln Gas Coal Co.
1928	100	¾–0	Link-Belt Baum jig	Hazlewood[a]	Jones and Laughlin Steel Corp.
1928	650	4 –0	Rheolaveur	Champion No. 1	Pittsburgh Coal Co.
1929	50	1 –0	American air tables	Humphreys	Humphreys Coal and Coke Co.
1929	700	4 –0	Rheolaveur	Nemacolin	Buckeye Coal Co.
1929	200	4 –0	Peale-Davis air tables	Brier Hill[d]	Deep Vein Consol. Coal Co.
1929	120	4 – ⅛	Menzies hydroseparators		Canonsburg Coal Co.
1929	450	4 –0	Rheolaveur	Champion No. 4	Pittsburgh Coal Co.
1930	450	4 –0	Rheolaveur	Champion No. 5	Pittsburgh Coal Co.
1930	140	⅝–0	American air tables	Clyde No. 3	W. J. Rainey, Inc.
1931	350	2 –0	Peale-Davis air tables and Link-Belt Baum jig	Hannastown	Jamison Coal and Coke Co.
1931	500	4½– ⅜	Chance Cone	Coverdale No. 8	Pittsburgh Terminal Coal Co.
1931	300	4 – ½	Menzies hydroseparators	Hutchinson	Westmoreland Coal Co.
1931	800	4 –0	Rheolaveur	Clairton[a]	Carnegie-Illinois Steel Co.
1933	300	4 – ¼	Chance Cone	Avella	Acme Coal Cleaning Co.
1933	100	4 – 5/16	Jeffrey Baum jig	Naomi	Hillman Coal and Coke Co.
1933 1937	400	4 –0	Rheolaveur and Stump airflows	Champion No. 6[e]	Pittsburgh Coal Co.
1936	100	4 –¾	Menzies hydroseparator	Champion No. 2	Pittsburgh Coal Co.
1937	350	4 –0	Link-Belt Baum jigs	Isabella	Weirton Coal Co.
1938	275	6 – ¼	Chance cone	Magee	Westmoreland Coal Co.
1938	400	4 –0	Menzies hydros and Stump air flows	Mather	Mather Collieries Co.
1938	15	⅜–0	Plato wet table	Atlantic No. 2	Atlantic Crushed Coke Co.
1939	75	4 –0	Link-Belt Baum jig	Isabella[f]	Weirton Coal Co.
1939	75	¾–0	Stump air flows	Graff No. 1 to No. 2	Westmoreland Mining Co.
1940	240	4 – ⅜	Rheolaveur	Hazelwood[a,g]	Jones and Laughlin Steel Corp.
1941	160	⅜–0	Rheolaveur	Hazelwood[a,h]	Jones and Laughlin Steel Corp.
1941	350	5 –1	Menzies hydroseparators	Warwick	Duquesne Light Co.
1941	325	4 – ⅜	Menzies hydroseparators and and hydrotator	National No. 3	National Mining Co.
1943	500	4 –0	Menzies hydroseparators and hydrotator	Crucible	Crucible Fuel Co.
1943	700	4 –0	Jeffrey Baum jig and hydrotator	Clyde	Republic Steel Corp.
1943	120	4 – ¾	Jeffrey diaphragm jig	Graff No. 1 and No. 2	Westmoreland Mining Co.

[a] Located at coke plant.
[b] Closed down about 1943.
[c] Closed down about 1939.
[d] Closed down about 1933.
[e] Located in Ohio but treating coal from Pennsylvania only.
[f] Treating mine rock and middlings from main washery, recovering steam coal.
[g] Replaced installation made in 1928.
[h] Addition to 1940 installation.

One method of securing uniformity is to mine coal from various sections or places of a mine at rates controlled to keep the analyses of the average coal below certain predetermined limits. This method involves the regular sampling of all working faces and is particularly applicable to mines operating in so-called "spotty" coal. The mine cars may or may not be controlled in their sequence of dumping. Some mines have storage bins ahead of the cleaning plant primarily for purposes of securing a uniform feed to the cleaning units. These bins also serve to some extent as mixing bins.

Other mines have bins designed especially to produce good mixing, in which the raw coal is distributed in successive layers along the entire length of the bin, while the coal is removed at several points from the bottom of the bin. The use of bins, either for storage ahead of the preparation plant or for the specific purpose of mixing the raw coal, has a beneficial effect by reducing variability of the raw coal. One mine in the Thin-Vein Yough group has three 1000-ton circular bins operating in parallel for storing the minus 4-in. raw coal ahead of the cleaning plant. In order to determine the reduction in the variability that was being obtained by the use of these bins, a series of individual increment samples were collected before the minus 4-in. raw coal entered and after it passed through the bin. These results are shown in Table 16 and indicate the mixing effect obtained by reducing the range in the ash of the individual increment samples. "Before the bins" the range was 13.9 per cent ash and "after the bins" the range was only 4.6 per cent ash. It is apparent that the storage of raw coal produced greater uniformity and greatly reduced the fluctuation in ash content of the coal to the cleaning plant.

As a corollary to controlling output by analyses, the practice of mixing or blending mine cars from various sections, according to their analyses, greatly increased the uniformity of the cleaned coal. One Greene County mine follows the system of "controlled output according to analyses," mixing of mine cars before dumping, and mixing by a specially designed mixing bin. Employing controlled dumping of mine cars according to their analyses of the raw coal is shown in Table 17, where it is seen that the range of sulphur in the cleaned coal sampled at the coke plant was reduced from 0.55 to 0.24 per cent sulphur. Under the two conditions of mixing the raw coal originated from the same sections of the mine during successive periods of operation and was washed in the same plant under identical conditions of operation.

The importance of uniformity in coke quality has been strongly emphasized in a recent publication[14] of the U. S. Bureau of Mines. This publication stresses the need for attaining uniformity in coke by attacking the problem at all possible points along the line from the selection and mining of coals to the blast furnace, and gives special attention to the increased uniformity that can be achieved by installing coal-cleaning plants.

MECHANICAL COAL CLEANING

In 1938, H. N. Eavenson made the statement that:

> Within the last 10 years cleaning coal by both dry and wet processes has become rather common at the largest plants in the Pittsburgh district, and now the district contains plants that for size and completeness are not surpassed anywhere. . . .
>
> The erection of these plants has been criticized by some as unnecessary, and as an unwanted and additional expense to the industry in this section. Neglecting the primary facts that increasing market demands for quality and uniformity and competition with better quality Southern coals made such plants necessary, the critics also overlooked the fact that much of the earliest coal washing in this country was done in the Pittsburgh district. In 1874 jigs were in operation at the Larimer coke plant, owned by Carnegie and Company, washing the slack coal shipped there from the Penn and Westmoreland mines, and in 1875 washers were in operation at Ellrod station and at two plants at Shaner station. In 1880 there were 13 plants washing coal in Allegheny, Fayette, and Westmoreland Counties, and an additional one was built at Latrobe in 1881.

Eavenson describes the types of washers in use in these plants in 1880, none of which would be generally known today. He adds:

> The use of these plants was gradually discontinued, and for many years, from about 1900 to 1926, little coal was washed in the entire area.

Table 18 is a tabulation of the cleaning plants treating Pittsburgh-seam coal that have been erected since 1912. In 1927, Pittsburgh Coal Co. built its first large cleaning plant and followed with three more from 1928 to 1930. Several of the steel companies also erected plants at coke plants or mines in the district from 1928 to 1931. These installations pioneered a new cleaning-plant program for the district that has continued to the present. The cleaning equipment installed has been largely of new types imported from Europe or developed in this country since 1925. The rated hourly capacity of the plants shown in Table 18 is 10,495 tons. Making corrections for abandonments and replacements, the net hourly capacity today is 9800 tons. Based on 2000 hr. operation per year, these plants could treat 19,600,000 tons annually. As many of the larger plants operate more than 2000 hr. per year, the annual input now probably exceeds 20,000,000 tons. One of these large plants, which operates 6000 hr. per year, has actually handled a little more than 5,000,000 tons during a 12-month period.

The authors have been unable to find any statistics giving directly the tonnage or proportion of Pittsburgh-seam coal that has been mechanically cleaned. The annual capacity of the cleaning plants listed in Table 18 when compared with the production of coal "probably from the Pittsburgh seam" given in Table 1 indicates that at least 25 per cent of the output is mechanically cleaned. Another approach is afforded by the data furnished in Minerals Yearbook covering the tonnage of mechanically cleaned or "washed" coal used in the manufacture of by-product and beehive coke. Tables 19 and 20 furnish this information for the years 1925 to 1942 inclusive.

TABLE 19.—*Washed[a] and Unwashed Coal Charged in By-product Coke Ovens*, 1,000 *Net Tons*[b]

Year	By-product Ovens								
	Total U.S.A.			New York, Ohio, Pennsylvania, and West Virginia[c]			Pennsylvania[d]		
	Total	Washed	Percentage Washed	Total	Washed	Percentage Washed	Total	Washed	Percentage Washed
1925	57,110	8,991	15.7	29,617	1,676	5.7	14,680	1,277	8.7
1926	63.647	10,007	15.7	33.441	2,246	6.7	16,895	1,725	10.2
1927	63,240	8,866	14.0	34.342	1,568	4.6	16,691	1,568	9.4
1928	70,166	10,580	15.1	39.259	3,156	8.0	20,326	2,464	12.1
1929	76,759	11,014	14.3	42,201	2,899	6.9	21,644	2,723	12.6
1930	65,521	10,905	16.6	35.568	3,768	10.6	18,776	2,137	11.4
1931	46,846	8,253	17.6	23,996	3,687	15.4	11,250	2,190	19.5
1932	30,887	6,717	21.7	15.394	4,170	27.1	6,115	2,916	47.7
1933	38,681	8,333	21.5	20,951	5,331	25.4	9,181	3,801	41.4
1934	44.343	9,464	21.8	24,029	5,625	23.4	10.083	3,890	38.6
1935	49,046	11,692	23.8	27,254	7,870	28.9	11,908	5,775	48.5
1936	63,244	15,342	24.2	36,297	9,741	26.8	18,380	7,474	40.7
1937	69.575	17,257	24.8	38,783	10.714	27.6	19,907	8,327	41.8
1938	45,266	13,523	29.9	23,135	7,972	34.5	10,400	5,825	56.0
1939	61,216	15,869	25.9	33,257	9,156	27.5	16,000	6,323	39.5
1940	76,583	19,667	25.7	42,641	11,219	26.3	21,719	8,134	37.5
1941	82,609	20,672	25.0	45.849	12,971	28.3	22,850	8,301	36.3
1942	87,974	23,453	26.7	48,381	14,408	29.8	23,889	9,956	41.7

[a] "Washed" refers to coal mechanically cleaned by either wet or pneumatic methods.
[b] Minerals Yearbook, U. S. Bur. Mines.
[c] From 69 to 75 per cent of the coal used was produced in Pennsylvania.
[d] From 85 to 90 per cent of the coal used was produced in Pennsylvania.

These data do not disclose the tonnage of coal from the Pittsburgh seam, but do give that from Pennsylvania, of which the Pittsburgh seam furnishes such a very large proportion. In Table 19 the large increase in the proportion of "washed coal" used in the manufacture of by-product coke in ovens in New York, Ohio, Pennsylvania, and West Virginia and in Pennsylvania only, undoubtedly reflects the increasing output of the coal-cleaning plants installed for cleaning Pittsburgh-seam coal given in Table 18.

TABLE 20.—*Washed*[a] *and Unwashed Coal Charged in Beehive Ovens in Pennsylvania*
NET TONS[b]

Year	Total	Washed	Percentage Washed
1925	14,572,282	1,690,751	11.6
1926	16,344,439	1,880,863	11.5
1927	8,729,862	848,799	9.7
1928	5,266,357	659,616	12.5
1929	8,195,528	1,036,799	12.7
1930	3,018,669	561,404	18.6
1931	1,309,929	242,352	18.5
1932	777,778	118,531	15.2
1933	1,052,491	128,665	12.2
1934	1,124,476	333,681	29.7
1935	881,499	171,098	19.4
1936	1,879,187	585,908	31.2
1937	3,906,754	730,707	18.7
1938	765,253	333,686	43.6
1939	1,775,307	382,297	21.5
1940	3,997,312	576,372	14.4
1941	9,196,363	1,096,784	11.9
1942	11,292,336	1,448,255	12.8

[a] "Washed" refers to coal mechanically cleaned either by wet or pneumatic methods.
[b] Minerals Yearbook, U. S. Bur. Mines.

Regarding the data on "washed coal" used in the manufacture of beehive coke in Pennsylvania (Table 20), this can quite safely be taken as representing Pittsburgh-seam coal, as the data in Table 3 show that virtually all coal so used comes from this seam. Table 20 shows that the tonnage and proportion of washed coal from the Pittsburgh seam coked in beehive ovens has fluctuated considerably between 1925 and 1942, but at present the proportion is about the same as in 1925 and 1926 while the tonnage is somewhat less. Table 19, covering the coal used in by-product ovens, shows a very different situation; the tonnage and proportion of "washed" coal has risen very considerably from 1928 to 1942.

From the information in Tables 19 and 20, it may be inferred that more than one half of the some 20,000,000 tons of mechanically cleaned Pittsburgh-seam coal annually produced in recent years has been used in the manufacture of coke.

ANALYTICAL RESULTS OBTAINED BY MECHANICAL CLEANING

The foregoing description indicates that the results obtainable by mechanical cleaning will be considerably affected by the seam characteristics and by the method of mining the coal, which largely determines the amount of higher ash and sulphur top and bottom coal included in the run-of-mine output. The method of mining also affects the amount of slate from above the coal and fire clay from below the seam that is loaded. Generally, the two latter impurities can be almost completely removed by efficient mechanical cleaning equipment, except in extremely fine sizes. Since fire clay and draw slate readily break into the extreme fines, they often create difficulties in their removal, because the fine impurities cannot be removed by air cleaning, and with wet cleaning slimes are produced from these materials and contaminate the water system.

The type of equipment used has some effect upon cleaning results and efficiencies. When mechanical cleaning is used the general practice is to wet-wash the coarse coal; that is, from ⅜ to 5 in. in size. On the sizes below ⅜ in., wet cleaning has been more generally used, but where the greatest improvement in the ash and sulphur of the fines is not imperative, air cleaning is often used. So far, most cleaning plants have been installed to produce only clean coal and refuse, but several produce a third product—a secondary coal of somewhat lower quality than the primary clean coal. This latter practice probably will be adopted more frequently in the future, to make possible a greater reduction in the sulphur of the primary clean coal that is to be used for cokemaking.

Typical performance of various types of mechanical cleaning plants handling coals from the several groups are given in Tables 21, 22 and 23. No attempt has been made to show the tonnage recoveries of the various plants, as it would complicate the presentation of the data. In all cases, the results cover the making of only two products, clean coal and refuse. Unless otherwise noted in the tables, the actual recovery of clean coal, dry basis, is equal to or slightly greater than the amount of raw coal floating at the specific gravity noted. The method of mining the raw coal in each case is indicated. Where the coal is entirely or almost entirely hand-loaded, the amount of refuse removed is usually from 3 to 6 per cent of the input tonnage.

TABLE 21.—*Typical Cleaning-plant Performance, Panhandle Group*

Mine or cleaning plant	Q		R		S		T	
Ash or sulphur, per cent	Ash	Sulphur	Ash	Sulphur	Ash	Sulphur	Ash	Sulphur
Feed coal[a]	11.4	1.80	9.1	2.05	9.9	2.70	10.7	3.35
Feed coal, float 1.60 sp. gr.[b]	7.1	1.55	6.4	1.75	6.7	1.90	6.7	2.45
Clean coal	7.6	1.67	6.7	1.82	7.0	1.96	7.1	2.51
Mark-up in analysis clean coal over feed coal, float 1.60 sp. gr	0.5	0.12	0.3	0.07	0.3	0.06	0.4	0.06
Reduction in analysis of feed coal to clean coal as percentage of feed-coal analysis	37.7	7.2	26.4	11.2	29.3	27.4	33.6	25.1

[a] 4-in. to 0 from run-of-mine that had been hand-loaded, with only 4 to 6 in. of bottom coal unmined. Coal cleaned by wet system.

[b] 4-in. to 48-mesh only.

TABLE 22.—*Typical Cleaning-plant Performance, Thick-vein Youghiogheny and Connellsville Groups*

Mine or cleaning plant	U[a]		V[b]		W[b]		W[b]		W[c]		BB[e]	
Ash or sulphur, per cent	Ash	Sulphur	Ash	Sulphur	Ash	Sulphur	Ash	Sulphur	Ash	Sulphur	Ash	Sulphur
Feed coal	8.06	1.24	8.16	1.31	9.01	1.25	8.88	1.37	10.39	1.57	14.8	1.11
Feed coal, float 1.60 sp. gr.	5.44[d]	1.08[d]									6.8[d]	1.04[d]
Feed coal, float 1.55 sp. gr.			5.59[d]	1.09[d]	6.75	1.05	6.66	1.09	6.77	1.16		
Clean coal	6.44	1.13	6.21	1.14	7.16	1.09	7.02	1.13	7.08	1.19	7.1	1.02
Mark-up in analysis clean coal over feed coal float at sp. gr. given	1.00	0.05	0.62	0.05	0.41	0.04	0.36	0.04	0.31	0.03	0.3	−0.02
Reduction in analysis of feed coal to clean coal as percentage of feed coal	20.1	8.9	23.9	13.0	20.5	12.8	20.9	17.5	31.9	24.2	52.0	8.1

[a] 4-in. to 0 from run-of-mine, hand-loaded with selective mining: i.e., top and bottom coal left unmined; 4 to ⅜-in. wet-cleaned; ⅜-in. to 0 air-cleaned.

[b] Run-of-mine crushed to minus 4-in.; hand-loaded with selective mining. All wet-cleaned.

[c] Same as *b* except that a small percentage of machine-loaded Greene County coal was included. All wet-cleaned.

[d] 4-in. to 48-mesh only.

[e] 4-in. to 0 from run-of-mine machine-loaded. All sizes wet-cleaned.

As the proportion of mechanically loaded coal increases, the amount of reject increases, and with all or nearly all mechanical loading it will vary from 8 to 30 per cent in all districts. This large increase in refuse content with mechanical loading comes primarily from added slate and fire clay from above and below the seam and from the inclusion of all the bearing-in bands.

Panhandle Group

The data of Table 21 give the results when treating only hand-loaded coal, 4-in. to 0 size, by a wet cleaning system. The percentage reductions in ash are considerable and vary from 26 to 38 per cent, being greater with the higher ash feed coals. It will be noted that the mark-up in ash analysis of the clean coal over the feed coal, 4-in. to 48-mesh size floating at 1.60 sp. gr., ranges from only 0.3 to 0.5 per cent ash. The reductions in sulphur are not so great, varying from 7 to 27 per cent, and are generally greater with the feed coals of higher sulphur. The sulphur analyses of the clean coals of Table 21 are quite high for coals used for making coke, but no special efforts were being made to obtain the ultimate in sulphur reduction, as these coals were not being used for cokemaking.

TABLE 23.—*Typical Cleaning-plant Performance, Greene County District*

Mine or cleaning plant	X[a]		Y[a]		Z[b]		Z[b]		Z[c]		Z[c]	
Ash or sulphur, per cent	Ash	Sulphur	Ash	Sulphur	Ash	Sulphur	Ash	Sulphur	Ash	Sulphur	Ash	Sulphur
Feed coal	8.40	1.55	10.14	1.70	9.43	1.64	9.50	1.65	12.81	1.67	14.03	1.90
Feed coal, float 1.60 sp. gr.[d]	5.53	1.25	7.54	1.34								
Feed coal, float 1.55 sp. gr.[d]					6.28	1.22						
Feed coal, float 1.50 sp. gr.[d]							6.56	1.24	7.17	1.33	6.83	1.36
Clean coal	6.36	1.32	8.40	1.47	6.80	1.26	6.83	1.29	7.49	1.34	7.56	1.45
Mark-up in analysis, clean coal over feed coal float at sp. gr. given	0.83	0.07	0.86	0.13	0.52	0.04	0.27	0.05	0.32	0.01	0.73	0.09
Reduction in analysis of feed coal to clean coal as percentage of feed coal	24.3	14.8	17.2	13.5	27.9	23.2	28.1	21.8	41.5	19.8	46.1	23.7

[a] 4-in. to 0 from run-of-mine. Hand-loaded with selective mining: i.e., top and bottom coal left unmined; 4 to ⅜-in. wet-cleaned; ⅜-in. to 0 air-cleaned.

[b] Run-of-mine crushed to minus 4-in.; hand-loaded with selective mining. All wet-cleaned.

[c] Run-of-mine crushed to minus 4-in. A large percentage of machine-loaded coal, where the bottom coal and more of the top coal than under *b* is mined. All wet-cleaned.

[d] 4-in. to 48-mesh.

Although data on the cleaning of mechanically loaded coal from the Panhandle group are not given, it is known that the changes in ash content resulting from washing are somewhat different than with hand-loaded coal. Owing to the much higher ash content of the feed coal due to free impurities, the reduction in ash will be greater. Mechanical loading tends to raise the mark-up in ash analysis over hand-loaded coal, owing to a greater proportion of extreme fines having an ash content that cannot be thoroughly cleaned.

The effect of mechanical loading on the removal of sulphur by cleaning cannot be reduced to general statements. A knowledge of the local mining conditions is essential for any predictions.

Thick-Vein Yough and Connellsville Groups

The data of Tables 21 and 22 indicate that hand-loaded mines of the Thick-Vein and Connellsville groups produced a lower ash feed coal than the hand-loaded mines of the Panhandle group. This is due primarily to leaving top coal as well as bottom coal unmined. As a result, the reduction in ash in the Thick-Vein and Connellsville groups is not so great as with the Panhandle group, being about 20 per cent. The data on plant W show that the mark-up in ash analysis of clean coal over feed coal floating at 1.55 sp. gr. has been held to a low figure, i.e., 0.3 to 0.4 per cent ash.

The sulphur content of hand-loaded coals in Table 22 is much lower than that of the Panhandle coals and the reduction in sulphur analysis is lower, being from 9 to 17 per cent. The reduction of 24 per cent shown by *W*(*c*) (Greene Co.) is largely due to higher sulphur coal in the feed, which was brought about by the change from hand to machine loading.

The results for a mechanically loaded coal are given under mine BB. A very great reduction in ash, 52 per cent, is shown, and the mark-up in ash is extremely low for such a high-ash feed coal. The sulphur content of the feed coal is low and consequently the reduction in sulphur is nominal. The negative mark-up in sulphur analysis and low mark-up in ash analysis are partly due to the flowsheet of the cleaning plant, which provides for the crushing of a substantial amount of middlings, which are rewashed.

Greene County Group

The data on Table 23 show results with both hand and machine-loaded feed coals, and with two systems of cleaning. In Table 23, results are shown for mine Z for both hand loading with selective mining (*b*) and for a high percentage of machine-loaded coal (*c*).

The two sets of data for Z(*c*) show higher ash in the feed, owing to machine loading of a large proportion of the coal. In one of these cases the feed-coal sulphur is about the average of that of the hand-loaded coals, while the other shows 1.90 per cent sulphur. The sulphur content of the feed coal, float 1.50 sp. gr., of the machine-loaded coal is higher than that of any of the hand-loaded coals. The reductions in ash are similar to those shown in Tables 21 and 22 with feed coals of similar ash content, and the machine-loaded coal shows a much greater reduction than the hand-loaded.

Like the Panhandle mines, the Greene County mines produce coal higher in sulphur content than the Yough and the Connellsville groups. The Greene County coals so far cleaned tend to have a lower sulphur content than the Panhandle group and, for the same feed-coal sulphur analysis, show a lower sulphur content in the feed-coal float at 1.50 to 1.60 sp. gr. For this reason, the reduction in sulphur analysis obtained with these Greene County coals is greater than with Panhandle coals of the same feed-coal sulphur content.

The coals wet-cleaned, mine Z, show lower mark-up in ash and sulphur analyses than the coals cleaned by combined wet and air systems, mines X and Y (Table 23). The results for the machine-loaded coal of mine Z, with a feed-coal analysis of 14 per cent ash and 1.90 per cent sulphur, show the higher mark-up in analysis caused by the higher impurity content of the 48-mesh to 0 feed coal that is only partly cleaned, and also by the sliming of the machine-loaded impurities.

The reductions in analysis obtained by coal cleaning are largely determined by the spread in analysis between the feed coal and that portion of it floating at the specific gravity of separation. This is shown by the data on mines X and Y. Mine X shows a spread of 2.87 per cent ash, or 34 per cent of the feed-coal analysis, while mine Y shows a spread of 2.60 per cent ash, or 26 per cent of the feed-coal analysis. The actual ash reduction obtained by an identical type of cleaning plant are 24 and 17 per cent, respectively.

Fig. 7 shows the improvement in ash and sulphur analysis effected by two cleaning plants, one handling Connellsville district coal and the other Greene County coal. The performance has been plotted against the ash and sulphur content of the feed coal. The data for each plant cover long periods of time and upward of several millions of tons of coal. The Connellsville coal was hand-loaded with selective mining except where the data show the highest ash and sulphur in the feed coal, in which there was included a small proportion of machine-loaded coal from Greene County. With the straight Greene County coal there was a gradual change from hand loading to more than 80 per cent mechanical loading. The information shown by

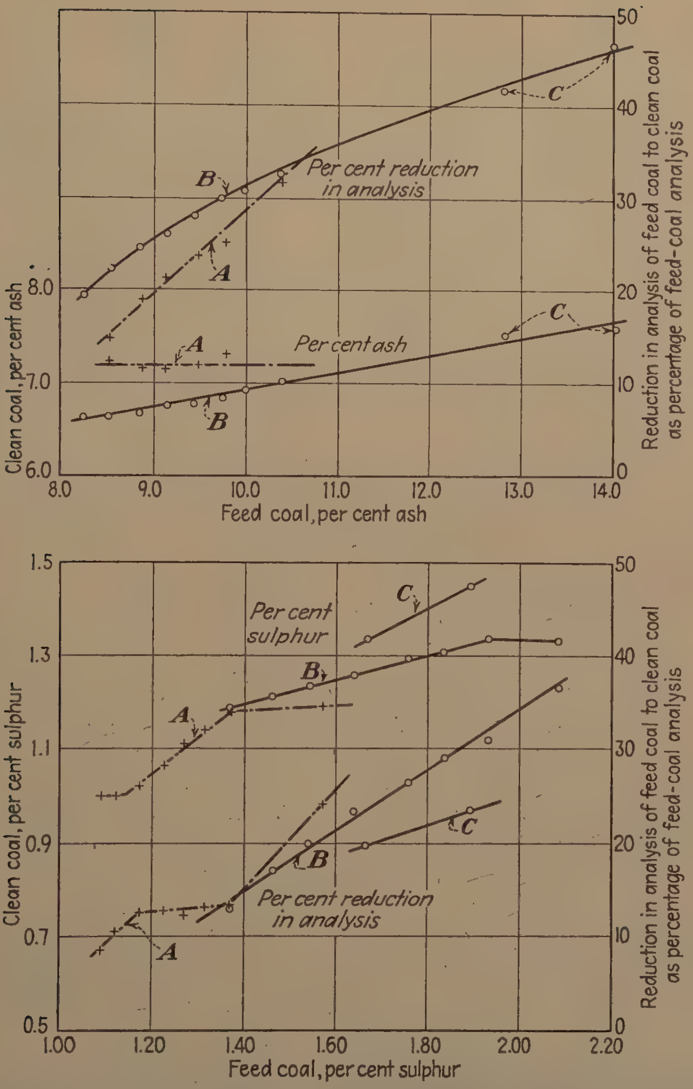


FIG. 7.—PERFORMANCE OF WET-CLEANING PLANTS TREATING RUN-OF-MINE COAL CRUSHED MINUS 4-INCH, ASH AND SULPHUR REDUCTION WITH VARYING QUALITIES OF FEED COAL.

A. Connellsville district, hand-loaded run-of-mine with selective mining.

B. Greene County, mostly hand-loaded run-of-mine with selective mining, but including some machine-loaded coal.

C. Greene County, over 80 per cent machine-loaded run-of-mine with the bottom coal and more of the top coal than under B loaded.

Fig. 7 is different from the data of Tables 22 and 23, but is in substantial agreement with the latter, bringing out additional relationships.

With the Connellsville district coals, the ash content of the clean coal remained practically constant as the ash content of the feed coal increased, while the sulphur analysis of the clean coal tended to increase as the sulphur content of the feed coal increased. This situation was caused by the inherent characteristics of this coal; i.e., the ash of the feed float coal remained substantially constant, with little or no increase in the ash of the extreme fines, while its sulphur content increased as the feed-coal sulphur analysis increased.

The data on the Greene County coal covers two mining conditions, which are indicated as B and C. The B data are from a period when a gradual change was being made from hand loading with selective mining to more than half of the coal being loaded by machines. Both ash and sulphur analysis of the clean coal increased slightly as the feed-coal analysis of each increased, although the percentage reduction in these analyses also increased at the same time.

A comparison of the C data, where over 80 per cent of production was machine-loaded, with the B data shows that the ash analyses and reduction in ash of the former fall on extensions of the curves for the latter. The sulphur curves show that the C data do not fall on the B-data curve, and the machine-loaded coal represented by the C data shows a definitely lower sulphur reduction. These conditions of increased ash and sulphur have been accounted for by an increase in the ash and sulphur content of the feed float coal; they are caused by loading of a greater proportion of top coal, and particularly of bottom coal, as the mechanically loaded tonnage increased. The organic sulphur of these coals is higher than the rest of the seam, and this form of sulphur cannot be removed by any of the known cleaning processes. Changes in analysis of the breast coal of the seam also occurred because of changes in locations being mined. This latter cause probably had a greater effect upon sulphur than upon the ash.

Reduction in Content of Various Forms of Sulphur Effected by Washing

In Table 24 are given data covering the reduction in the amount of pyritic, sulphate and organic sulphur obtained by wet cleaning plants treating coals from the Panhandle, Thin-Vein Yough, Thick-Vein Yough and Greene County districts. These data show that only the pyritic and sulphate sulphur is reduced by washing and, as might be expected, the organic sulphur analysis tends to be increased.

The results shown in Tables 21 to 24, inclusive, and Fig. 7 must not be taken as the only results that can be attained with these plants, for the specific gravity at which separation of refuse can be made is subject to a reasonable degree of control. However, these results may be considered as representing the economic separations under the conditions in force at the time and with the plant in use.

Reduction in Phosphorus Content Obtained by Mechanical Cleaning

The effect of mechanical cleaning on phosphorus analysis is indicated by the following examples, which are rather fragmentary because information is not readily available on this subject.

The following data on a run-of-mine coal from a mine in the Thick-Vein group, which was cleaned by hand-picking the lump, wet-cleaning the 3½ to ⅜-in. size and dry-cleaning the ⅜-in. to 0 size, confirms the information given earlier on the distribution of phosphorus in the various benches of the Pittsburgh seam; i.e., that the coarser sizes, coming largely from the upper portion of the seam, will carry the higher phosphorus content. The analyses of the various sizes of clean coal produced by this method of cleaning were:

CLEAN COAL SIZES	PHOSPHORUS, PER CENT
+4-in. lump	0.021
4 to 2-in. round	0.013
2 to 1⅛-in.	0.010
1⅛ to ⅜-in.	0.008
⅜-in. to 0	0.011
Assembled mine-run	0.011

In this case no phosphorus analyses were made of the raw coal.

At a Greene County property, phosphorus analyses on raw coal and clean coal are available on daily samples for a period of 220 days. The run-of-mine coal was crushed through a 4-in. round screen after passing through a large blending bin, and all sizes were wet-washed:

Greene County, 4-in. to 0 Coal 220 Daily Analyses	Phosphorus, Per Cent	
	Raw Coal	Washed Coal
Maximum analysis	0.0170	0.0140
Minimum analysis	0.0120	0.0090
Range in analysis	0.0050	0.0050
Average analysis	0.0150	0.0114
Reduction in analysis of raw coal to clean coal as percentage of raw coal analysis	24.0	

The percentage reduction in phosphorus analysis shown by these data lies about half way between the reductions in ash analysis and in sulphur analysis shown by the same plant.

TABLE 24.—*Reduction in Content of Various Forms of Sulphur Effected by Wet-cleaning Plants Treating Pittsburgh-seam Coal*

Size of Coal	District[a]	Sulphur, Dry, Per Cent						Reduction of Sulphur Analysis, Per Cent		
		Feed Coal			Clean Coal					
		Pyritic + Sulphate	Organic	Total	Pyritic + Sulphate	Organic	Total	Pyritic + Sulphate	Organic	Total
4-in. to 0	Panhandle	1.13	1.02	2.15	0.98	1.00	1.98	13.3	2.0	8.0
4-in. to 0	Thin-Vein Yough	0.39	0.73	1.12	0.26	0.75	1.01	33.3	−2.7	9.8
4-in. to 0	Thick-Vein Yough	0.40	0.68	1.08	0.23	0.75	0.98	42.5	−9.3	9.3
4-in. to 0	Greene County	0.67	0.93	1.60	0.37	0.93	1.30	44.8	0.0	18.8
5/16-in. to 0	Greene County	0.89	0.93	1.82	0.45	0.93	1.38	49.4	0.0	24.2

[a] These results are from one mine only in each district. They are indicative only and do not necessarily represent an average picture for the district.

UNIFORMITY OF MECHANICALLY CLEANED COAL

In addition to improving the average qualities of a coal by removing impurities, mechanical cleaning materially improves its uniformity; i.e., reduces its variability. Much has been published[15,16] on the uniformity in ash and sulphur analysis of mechanically cleaned Pittsburgh-seam coal. It has been stated[16] that: "The uniformity (of the cleaned coal as to ash content) is materially increased, i.e., the variability is decreased to a value one half to less than one-sixth that of the raw (feed) coal; . . ." This statement was based on the records of plants treating Pittsburgh-seam coal. In the same paper data are given for Pittsburgh-seam coal, showing that the variability as to sulphur content of the cleaned coal was nearly one half that of the feed coal.

BENEFITS OF WASHING PITTSBURGH-SEAM COKING COALS

The economic benefits in the manufacture of pig iron resulting from the washing of Pittsburgh-seam coking coals have been ably presented in the classic paper by C. D. King.[5] He gives the comparison between coke made from unwashed and washed coal from the same mines in the Connellsville district as are shown on Table 25, and says:

TABLE 25.—*Amount and Variation of Ash and Sulphur in Coke Made from Unwashed and Washed Pittsburgh-seam Coal, Connellsville District*[5]

PER CENT

Variation for Any Single Day Based on Periodic Daily Tests Using Half-hour Samples for Four-months Period	Furnace Coke Produced from					
	Unwashed Coal			Washed Coal		
	Maximum	Minimum	Range	Maximum	Minimum	Range
Ash....	14.05	11.08	2.97	11.04	10.08	0.96
Sulphur	1.08	0.82	0.26	0.97	0.80	0.17

Variation Based on Daily Average Analysis for Every Day of the Four-months Period	Unwashed Coal	Washed Coal
Average ash..................	11.87	10.36
Average day to day variation in ash..........................	0.40	0.14
Variations exceeding 0.30 per cent ash..........................	50.00	9.50
Variations exceeding 0.50 per cent ash..........................	31.50	0.00
Variations exceeding 1.00 per cent ash..........................	6.70	0.00

> The four months test performed at the Clairton by-product plant to determine the effect of washing coking coals indicated that a marked improvement can be expected in ash, sulphur, physical properties and uniformity of the coke so produced.

In summarizing the results obtained in the blast furnaces with the cokes made from unwashed and washed coals, he says:

> The following advantages were obtained by the use of such (washed coal) coke.
>
> 1. Daily iron production was increased 63.3 tons, or equivalent to 8.1 per cent.
>
> 2. Net furnace coke consumption was decreased 149 lb., or equal to 7.8 per cent.
>
> 3. Flux consumption was decreased 198 lb., or equal to 19.9 per cent, and slag volume was reduced 15.2 per cent.
>
> 4. A pronounced improvement was obtained in the regularity of pig-iron analysis.
>
> In addition, an item of considerable importance is the fact that the increased yield of furnace coke of 3.0 per cent at the Clairton coke plant, combined with the subsequent reduction in coke requirements per ton of iron, means less mined coal is required per ton of iron under such conditions than when using coke from unwashed coals. In the former case, 1.375 net tons of mined coal were required and in the latter 1.501, or a difference equal to 8.4 per cent less mined coal needed when using washed coals.

SUMMARY AND CONCLUSIONS

The Pittsburgh seam of coal in Pennsylvania is of great importance to the economy of the United States, as it has in recent years contributed 12 to 15 per cent of the total bituminous tonnage and, of greater note, from 32 to 41 per cent of the tonnage of coal made into coke.

River transportation, because of low cost, is a dominant factor in the use of Pittsburgh-seam coal for cokemaking and in the steel industry, and more than 35 per cent of the 1941 production was loaded on the Monongahela River.

The reserves of the Pittsburgh seam in Pennsylvania of high-grade and established coking quality are being rapidly depleted. The more rigid specifications to be met in the manufacture of steels require more uniform raw materials for satisfactory costs, while mine mechanization tends to cause a lowering in the quality of coal, and, therefore, of coke.

Because of a variety of economic factors, no exact and complete specifications for coking coals have been accepted. Using sulphur content as an example, in the recent past 1.20 per cent has been the limit for Pittsburgh-seam coking coals, but today it is not rigidly applied. With coal having access to river transportation, about

1.50 per cent sulphur might be economically equivalent to 1.20 per cent sulphur for coal requiring rail transport.

Recoverable Pittsburgh-seam coal remaining in Pennsylvania as of Jan. 1, 1937, is estimated as follows:

1. Total, 6,500,000,000 tons, or about 100 years life at the recent production rate. Greene and Washington Counties, which are not fully developed, contain nearly 90 per cent of the reserves.

2. Coal tributary to river loading, 1,600,000,000 tons, or about 25 per cent of the total reserves.

3. Coal analyzing 1.50 per cent sulphur or less, 1,656,000,000 tons, or about 55 years production for cokemaking at the recent annual rate of 30,000,000 tons. If coking coal is limited to coal analyzing 1.20 per cent sulphur or less, the tonnage probably shrinks to about 600,000,000 tons, or only about 20 years life.

These statements are based on the assumption that the large reserves in Greene and southern Washington Counties, which are shown as not fully developed, will yield coal analyzing 1.50 per cent sulphur in amounts that will have little effect on the over-all life of these reserves. Such information as is available substantiates such an assumption.

The analytical and physical characteristics of the Pittsburgh seam in Pennsylvania have been summarized and discussed under five general groups: Panhandle, Thin-Vein Youghiogheny, Thick-Vein Youghiogheny, Connellsville, and Greene County (also southern Washington County). Maps have been prepared showing the locations of fairly well-defined areas of coals of several sulphur contents—1.20 per cent or less, 1.21 to 1.50 per cent, 1.51 to 1.80 per cent, and over 1.80 per cent—and their relation to these various groups.

Sulphur occurs in the organic, sulphate, and pyritic forms. The forms are shown to vary both within the seam by benches, by areas within a mining property, and by specific-gravity fractions. In the various benches organic sulphur is highest in the top and bottom coals and lowest in the top and bottom breasts or middle benches; it is low in the lighter gravity fractions and the least in the high-gravity fractions (over 2.0 sp. gr.). Pyritic sulphur is also the highest in the top and bottom coals and in respect to specific gravity it is the highest in the heavy-gravity fractions. The approximate range in the percentage of the various sulphur forms is indicated for each group. Organic sulphur is highest in the Panhandle and Greene County groups. Sulphate sulphur is low, usually amounting to 1 or 2 per cent of the total sulphur.

This study of the benches of the Pittsburgh seam in the various groups shows that selective mining can improve the quality of the production, mainly by leaving bottom and top coal unmined. Such methods applicable with hand loading cannot always be used with mechanical loading with mobile loading machines. Thus, and because of concentrated production with mechanical loading, its output often carries a higher and more irregular content of impurities, and usually requires mechanical cleaning for the production of marketable coal. In many cases, to obtain a satisfactory uniformity in analysis, particularly as to sulphur, additional practices must be employed, such as controlled output from areas of differing levels of quality with the dumping of mine cars controlled to blend these differing coals, blending or mixing bins and layer loading of output in railroad cars or barges.

Mechanical cleaning has been extensively used to clean coal from all groups, either hand or machine loaded, to standards of purity and uniformity as to ash content to satisfy all general market requirements, and especially the severe demands of the metallurgical coke industry. The beneficiation as to sulphur content is usually not so great and varies quite widely, dependent upon the qualities of the feed coal. Reductions in ash content of 20 to over 50 per cent of the feed-coal value are obtained, while with sulphur content the reductions range from 8 to 25 per cent. The removal of sulphur by mechanical cleaning is confined primarily to the pyritic form and the amount removable depends on the particle size of the pyrites, much being microscopically disseminated through the coal substance and thus not removable. Some coals have allowed a removal of 40 to 50 per cent of the pyritic sulphur by mechanical cleaning.

Mechanical cleaning adds a very considerable degree of uniformity to the ash and sulphur analyses of the coal.

The experience of one large steel company has been that only 1.375 net tons of raw coal, when washed, produces the coke necessary for one ton of pig iron, while 1.501 net tons of the same coal, when coked unwashed, is required for the same pig-iron production. This is equivalent to extending the life of their reserves 9.6 per cent.

The maximum economical reduction in ash and sulphur content and uniformity in these values will be obtained only by a coordination of the mining practices and the mechanical cleaning program.

It appears safe to predict that with proper coordination of mining practices and present mechanical cleaning processes, part of the large reserves in Greene County of higher sulphur content may be made available for the manufacture of metallurgical coke as follows:

Raw coals of 1.50 per cent sulphur or less can be reduced to about 1.20 per cent sulphur or less.

Raw coals of 1.51 to 1.80 per cent sulphur can be reduced to about 1.21 to 1.50 per cent sulphur.

Raw coals above 1.80 per cent sulphur can be reduced by about 20 per cent of their sulphur analysis.

These predictions can also be applied to the reserves in the southeastern quarter of Washington County.

ACKNOWLEDGMENT

In addition to the references cited, we wish to gratefully acknowledge the helpful criticism of Mr. H. N. Eavenson and to thank Mr. J. B. Morrow for permission to use information assembled by himself and the Pittsburgh Coal Co. The aid of Mr. Jack M. MacLachlan in preparation of the maps, seam diagrams, and assembly of data is also acknowledged.

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DISCUSSION

(*H. F. Hebley presiding*)

C. D. King,* Pittsburgh, Pa.—This paper is so complete in all respects that there is little that I can add. It constitutes an important reference and valuable contribution to our knowledge on the subject. I believe that we all feel that the better coals of the Pittsburgh district are tending to be a thing of the past. We are faced with poor coal, high in sulphur and high in ash. Mechanized mining, which is axiomatic under large-scale operations and present-day conditions, involves an increase in impurities. As I see the problem, there are two choices, and only two. We can accept the poorer coal and pay for it in lower blast-furnace production and higher operating cost, which essentially means a greater investment to produce equal amounts of pig iron; or we can try mixing, blending and beneficiating the coals, in order to obtain normal or greater production from blast furnaces. The second alternative also involves additional costs and investments. The experience indicates that it is the preferable course of action.

W. T. Brown,† Pittsburgh, Pa.—The city of Pittsburgh is known throughout the world as the "Steel City," but everyone in the steel industry knows that the only reason that Pittsburgh is a steel center is that low-cost, river-borne coal is available for the production of blast-furnace coke. Take cheap coal away from the steel industry in Pittsburgh and it could not compete with the plants on the Lakes.

The reserves of low-sulphur coals in the Pittsburgh district are being exhausted rapidly, as the authors point out. If the steel industry has to use coals containing 1.90 to 3.00 per cent sulphur in making blast-furnace coke, the production of the blast furnaces will be considerably reduced, particularly when the silica content of the iron ore becomes high—about 10 years hence. The desulphurization of pig iron is not practical with the low sulphur specifications for pig iron and steel. If the sulphur is reduced in the open-hearth furnaces, the capacity of each furnace will be reduced.

The place to reduce the sulphur is in the coal—any other method will increase the cost of steel and decrease production.

The maps and tables shown by Davis and Griffen show clearly the rapid depletion of the low-sulphur coal reserves and the large reserves of high-sulphur coals in the Panhandle and Greene County groups. The paper also shows the concentration of high sulphur in the top and bottom sections of the Pittsburgh bed, even in the low-sulphur sections of the seam.

The year of 1943 showed a decided trend to mechanical loading of bituminous coal and the construction and enlargement of many coal-cleaning plants to offset the increase in ash and sulphur due to mechanical loading.

The bituminous coal industry set a new all-time production record in 1943, by producing approximately 589,000,000 tons of coal (including approximately three million tons of lignite), in spite of strikes, work stoppages, labor troubles, absenteeism and the loss of nearly 60,000 experienced miners to the armed forces and defense plants.

The coal industry faces an immense task in trying to meet the nation's requirements of 625,000,000 tons of bituminous coal, which is the goal set for 1944. It will be necessary to install more and more mechanical loading equipment to take the place of manpower if the industry is going to meet its requirements.

Mechanical loading of coal increases the ash content at least 2 per cent, and even more at some mines, because there can be little or no preparation of the coal at the face when mechanical loaders are used. The sulphur content will also be increased at many mines by loading of the top and bottom sections of the coal bed, which often are high in sulphur. Formerly the experienced coal miner could hand-load clean, low-ash and sulphur coals, by means of proper preparation at the face.

The manpower shortage makes mechanical loading of coal a necessity; mechanical loading increases the ash and sulphur content; therefore coal cleaning is a necessity.

Therefore, the by-product coke industry must realize that the ash and sulphur content will be increased even in sections of the Pittsburgh seam that have in the past produced low-sulphur and low-ash coal.

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[†] Consulting Coal and Coke Engineer.

The coal operator asks: "Just what kind of coal does the coke-oven operator want, and why does he require such high-quality coal?" The coke-oven operator's requirements vary with the design of his coke ovens, temperature of carbonization and the various coals used in the mix charged to the coke ovens. In general he requires:

1. Uniformity.
2. Low ash; 6.5 per cent (coke produced from 6.5 per cent ash coal will contain from 8.8 to 9.2 per cent ash). (Coal ash × 1.35 to 1.40 = coke ash.)
3. Low sulphur; under 1.0 per cent. (But he can use coal with sulphur content up to 1.30 per cent, provided it is uniform. The sulphur content of the coke will be 80 to 85 per cent of the sulphur in the coal.)
4. Carbonization properties should be good, in order to produce uniform, blocky coke of good structure.
5. Expansion properties of the coal, when mixed with other coals charged to the coke ovens, should be neutral or slightly contracting, in order to prevent damage to the brick side walls of the coke ovens.
6. High-ash-fusion coal is of no advantage when used to produce blast-furnace coke but is desirable for domestic coke.

In the manufacture of blast-furnace coke in by-product coke ovens, it is necessary to use low-ash and low-sulphur coking coals, in order to produce coke that will increase iron and steel production. High-ash coke reduces iron production 5 per cent for each one per cent increase in ash content—over 9 per cent—and requires more coke per ton of iron.

High-sulphur coke requires more limestone, and more limestone means more coke and less iron-ore burden, which decreases amount of iron produced and requires more coke per ton of iron produced.

Coal cleaning will improve coking coals by uniformly reducing the ash and sulphur content, which in turn will improve blast-furnace coke and increase iron and steel production. Also less coking coal will be required to produce a ton of pig iron if the blast-furnace coke is improved by making it uniform in size and structure and uniformly low in ash and sulphur.

There is much low-sulphur coal wasted in the mine and an increase in sulphur in coals mined in the Pittsburgh bed, by cutting a certain distance from the top or bottom in selective mining to improve the sulphur content. The writer has discovered a method for detecting lenses and bands of pyritic sulphur in coal seams and lumps of coal that will prevent the waste of good coal and show the cutter exactly where to cut in any part of the seam. If the whole seam is mined, these high-sulphur lumps may be picked out on the picking table by this new method.

Davis and Griffen have shown that the ash in the high-sulphur coals in the Panhandle and Greene County groups may be reduced but that the high pyritic sulphur cannot be satisfactorily decreased by the present methods of coal cleaning. The writer has applied for a patent on a new process that will reduce the sulphur content of high-sulphur coals to produce a by-product quality coal. This new process is based on sound engineering principles and will be ready shortly for the coal industry.

The authors say that coal from the Pittsburgh seam in Pennsylvania will "make coke usable in the blast furnace. There may be variations in the yield of coke and in its physical properties which affect its value in the blast furnace, but these are usually of relatively minor importance." The coal in the Pittsburgh seam, whether in Pennsylvania, West Virginia or Maryland, varies, as does coal from any other seam, according to the geological pressure to which the coal has been subjected. Coal in the Pittsburgh district varies from 7.0 to 30.0 per cent contraction in the expansion tester, and further east, the greater the geological pressure and of course the lower the volatile and the greater the expansion. Therefore, coal that has an expansion of as much as 16.0 per cent is found in Maryland near the Allegheny escarpment. Much of the strip coal from the Pittsburgh seam makes poor coke and sometimes comes out of the coke oven like sand and gravel.

H. N. Eavenson,* Pittsburgh, Pa.—The data contained in this paper show how difficult it is to reduce the sulphur in Pittsburgh-seam coal, and the reasons why this is so. The amount of sulphur in coal to be coked that will be accepted at the ovens has increased greatly in recent years, and while some of this is due to war conditions, most coke producers are reconciled to the fact that coal below 1.0 per cent sulphur has gone forever, and that if the sulphur content can be held to 1.25 per cent in the raw coal, they will be fortunate. One coke plant, at least, is willing to accept coal 1.35 per cent or less, and one captive producer ships coal as high as 1.5 per cent, although the ore conditions make this high sulphur content possible.

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* Eavenson, Alford and Auchmuty.

It is definitely certain that in the future coals higher in sulphur must be used in cokemaking and cost considerations require the use of Pittsburgh-seam coal. Much study must be given to the necessity of keeping the amount of sulphur as low as possible and to all possible means of effecting this, and of course some sulphur must be removed after iron has been made. It is entirely probable that at some time in the near future the river transportation available will enable the regular use of some percentage of the Southern coals, which are very low in sulphur and ash, to blend with Pittsburgh-seam coal and to keep the ash and sulphur contents of the blend within more usable limits.

The effects of uniformity of product produced by washing are very well shown, and for coke show that the added cost is well worth it. It has been definitely established in practice that steam producers in the Pittsburgh district are not willing to pay more for washed than for raw Pittsburgh coal, and do not think the reduction in ash or the uniformity of product of any money value to them. From an economic viewpoint, it is unfortunate that a market cannot be created for a middlings product, which would allow better coal to be furnished for coke.

The maps and charts are extremely well made and present data not available to the public heretofore, and to only a few companies, and which is very valuable for reference purposes. The paper is a classic.

A. R. Powell,* Pittsburgh, Pa.—This excellent and comprehensive description of the Pittsburgh seam in Pennsylvania is particularly timely because of the rather radical changes in the nature of the reserves that are just now making themselves evident to a most unwelcome degree.

To sum up the whole matter, large reserves of this important coking coal still remain, but unfortunately these reserves are mostly at a distance from water transportation facilities and/or are high in sulphur content. I do not wish to touch on the transportation angle of the problem, but I shall think out loud for two or three minutes on the subject of sulphur, since this has been rather close to me for the last 30 years.

Whereas Nature has been kind to us in not mixing mineral matter so thoroughly with the pure coal substance but that most of it may be chopped off the coal by proper mining methods and coal beneficiation, she has not been so thoughtful in connection with sulphur. As the authors have shown, more than half of the sulphur, on the average, is chemically combined as organic sulphur with the coal substance itself, and still more is present as pyrites so finely disseminated throughout the coal substance that it might just as well be regarded as an intrinsic part of the pure coal as far as any possible separation is concerned. In other words, sulphur, for the most part, is like a dread disease that has penetrated to all of the tissues of the body and may be destroyed only by the complete destruction of the victim.

Since only the coarse particles of pyrites may be removed, and since these account for only a minor part of the sulphur in Pittsburgh-seam coal, sulphur reduction averages only about 20 per cent, even by the most modern washing methods. Fortunately, however, most of the up and down variations in sulphur content are due to coarse pyrites, so that washing has the effect of cutting off peaks and thereby reducing variability. This should serve to temper somewhat our disappointment over low sulphur removal, since a more uniform sulphur content in metallurgical coke, like uniform ash content, is of major importance in securing good results in the blast furnace.

When considering the sulphur problem, a very important fact to keep in mind is that only about 2 per cent of the sulphur in the raw coal finally appears in the pig iron. In other words, 98 per cent of the sulphur is eliminated along the way—say 20 per cent by coal washing, 40 per cent of the remainder during carbonization of the coal, and the residue, except for the 2 per cent that appears in the pig iron, passes out of the blast furnace in the slag.

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* Research Department, Koppers Company.

It would appear, in view of the growing sulphur problem in the Pittsburgh seam, that the time is ripe for comparative economic studies of various expedients to meet the situation. In the past the method used for acceptable sulphur practice in the blast furnace has been simple—merely confine production of metallurgical coal to the low-sulphur districts of the Pittsburgh seam. What other expedients must now be used to supplement the narrowing choice of coal reserves?

Selective mining, blending at the mines, and coal washing? Yes, most decidedly, for ash reduction and uniformity of quality, if for no other reason. Despite such measures the sulphur content of the coal still remains relatively high, however.

Methods have been proposed, and have been demonstrated on an experimental scale, which increase above the normal amount the elimination of sulphur during the coking process, or which remove sulphur from the finished coke. None of these methods are simple or cheap, and all involve some complications. Is the reduction of sulphur in the coke worth this expense and trouble? Or should the increased sulphur be allowed to enter the blast furnace and be handled there by some modifications in slag volume and composition? Or should the increased sulphur be allowed to enter the pig iron, and be followed by desulphurization in the ladle?

In other words, many expedients are available for removing sulphur all the way along the line from the raw coal in the mine up to the finished pig iron. All of these cost money and entail some inconvenience. Comparative economic studies of all of these methods are now very much in order. This would be an excellent topic for some future A.I.M.E. paper.

L. E. Young,* Pittsburgh, Pa.—The authors are to be congratulated on this splendid and timely paper. It is to be hoped that similar surveys will be made on the coals from other fields.

Permit me to direct attention first to a paper entitled "Control of Sulphur and Ash in Mine-run Metallurgical Coal," published as Bureau of Mines *Report of Investigations No.* 372. Another valuable contribution on underground preparation is chapter 7 of the volume on Coal Preparation,* which was the work of John D. Cooner and M. H. Forester.

The question is: "How can we improve the quality of the coal mined in southwestern Pennsylvania for the steel industry by special methods of cutting, drilling, shooting, and mechanical loading?" The salient points appear to be: (1) selective mining and (2) avoidance of excessive degradation.

Reference is made to cutting practice in both the Bureau of Mines investigation and the chapter in Coal Preparation. Normally, in bituminous coal mining, a single cut is made, either at the top or bottom, and the paper indicates that in a large part of this field top-cutting has been followed generally for the purpose of leaving up some of the high-sulphur top coal. When this coal is left in place, it serves, of course, as support for the drawslate. Where height is not adequate, this practice cannot be followed. There also are problems in regard to the sulphur in the bottom coal and in parts of the field sulphur in the bottom coal is a much more serious problem than sulphur in the top part of the seam. Usually several inches of bottom coal are left, whether the coal is top-cut or bottom-cut.

In the book on Coal Preparation, reference is made to the use of the cutting machine with a double cutter bar. To the best of my knowledge, the double cutter bar is not used in southwestern Pennsylvania at this time, although it is used extensively in central Pennsylvania. This type of machine has a built-in conveyor, which permits the cuttings to be loaded directly into a mine car as the cutting proceeds, or which makes it possible to throw the cuttings into the gob. By the use of this machine, a portion of the seam can be disposed of separately, or sometimes a superior grade of coal can be cut out and loaded separately.

In the Birmingham field, shelf mining is being practiced at the present time. By the use of a special loading machine the top part of the seam is loaded separately. After the upper part has been loaded, the lower bench is shot and the same loading machine loads the remaining coal. In some districts, the bottom bench of coal is mined in advance of the top bench, the latter being shot down after several cuts of the bottom bench have been removed. Various schemes may be suggested to mine benches separately, but most of these methods add substantially to the cost of mining and most of them are difficult of application when complete recovery of pillars is an objective.

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* Consulting Engineer.

* Seeley W. Mudd Series, A.I.M.E., 1943.

The problem of shearing and the proportion of fines in the cutting are subjects that have received a great deal of attention. In the matter of both wet and dry cleaning of coal, the reduction of the amount of extremely fine coal in the cuttings is of prime importance. Many studies have been made of cutting bits, the lacing of cutting chains, thickness of kerf, etc. The British probably have gone further than we have in developing mechanical devices for removing the cuttings from the kerf. A so-called "gumming" machine has been in use in Britain for some time and one American manufacturer has developed a "bug-dusting" machine which removes the cuttings promptly, so that they are not drawn back into the kerf and pulverized.

Engineers interested in the washing of coal and in the manufacture of coke are particularly concerned over the increase in fines incident to mechanical loading. It seems timely, therefore, that the mining men should review the entire subject of face preparation in mechanical loading with the hope that we may produce coarser coal, not necessarily lump coal, thus reducing the extremely fine material that becomes a burden in the washing and drying plants.

S. M. Cassidy,* Isabella, Pa.—Messrs. Davis and Griffen are to be congratulated upon the thoroughness with which they have covered their subject, particularly the cleaning data and the factors that influence proper washing to obtain both uniformity and quality in coals for blast-furnace coke.

While selective mining—both as to controlling the rate of mining from given high-sulphur and low-sulphur areas, and as to mining only selected benches in a seam, or as to gobbing or loading out separately certain partings or inferior coal—has been and is being done quite extensively to benefit the coal quality, its general use is open to serious question. There are coal seams and conditions where selective mining may be justified in particular cases, but this is less true of the Pittsburgh seam, where loading mechanically en masse has so many cost and practical advantages as to warrant the installation of proper facilities in the cleaning plant to really make it possible to handle the large amount of mine rock, partings, inferior coal and coal of intermediate grade, yet give the required uniformity and analysis in the coking coal.

It has too often been the practice to build cleaning plants for handling a fairly clean product from the mine, this being largely based on conceptions held over from hand loading, without realizing that to continue the practices into mechanized mining placed a terrific handicap on the new method.

By far the greatest cost at any mine is at the working face, consequently any practice that will simplify operations, lower cost, or permit the loading of a higher tonnage at this point can usually be shown to benefit the mine cost as a whole, despite the fact that the preparation plant and refuse-disposal facilities will require a larger investment and will have a higher cost per ton for their individual items.

It stands to reason, and can be proved in most cases, that it is far better and less costly to handle the mass of mixed coal-seam material in an outside plant, where adequate facilities can be provided, than to selectively mine, cut, gob, hold up or leave down the coal and impure layers at a multitude of scattered working places by a number of more or less portable machines operated by many men. The improvement in mine, safety and working conditions from taking the entire seam is usually appreciable, but the advantages of this are often not given their proper consideration.

Quality and uniformity of clean coal can be and are now being realized at certain mines where selective mining has been deliberately discarded. At these mines the tail does not wag the dog—mining is done to give best mining results and all following operations to and through the preparation plant are designed for and subordinate to the mining requirements. There are other mines where entire-seam mining is being gradually approached, not from choice originally but from force of circumstances and a gradual realization of advantages.

Mining from selected areas at such rates as to attempt control of quality has been practiced in many places, but this is too often overdone.

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* Manager, Weirton Coal Company.

Selective area mining, in the past usually to defer installation of blending bins and cleaning plants, is now proving very much to the detriment of certain operating mines and their cost per ton.

Elaborate layouts at the dumping and hoisting points, to permit shuttling and rearrangement of cars from various sections to blend high-sulphur and low-sulphur coal, are usually unnecessary, as the same results, or better, can be obtained by proper blending bins. Such arrangements are higher in both investment and operating cost nearly always than the use of blending bins. It can also be shown that blending at too many points tends to unblend in certain cases or at certain intervals.

Quality and uniformity of coke for blast furnaces is highly important, but this should be obtained in the best and most economical manner. If this can be furnished by really properly designed preparation plants, then all the agony and high cost of selective area mining, selective bench mining and elaborate but costly bottom layouts, are certainly unjustified, as has been proved by actual practice at a number of mines.

R. D. Hall,[*] New York, N. Y.—One company made it a practice to consign to the railroads the coal prepared by the washer in the morning and to send to metallurgical furnaces that prepared in the afternoon, thereby getting a better metallurgical coal because in the morning the washer received the undercuttings as well as some of the coal blasted from the Pittsburgh seam, and in the afternoon the product was all of the latter type, this being cleaner than the coal from the kerf.

H. Zeller,[†] Greensburg, Pa.—We agree with both the conclusions and estimates in the paper by Davis and Griffen.

We have been interested very much in the question of utilizing the Pittsburgh-seam coal in the undeveloped reserves in the western part of the state, and the greatest difficulty that we can see is the high percentage of organic sulphur, together with the very fine distribution of the pyritic sulphur. In order to get any sort of a sulphur reduction by cleaning because of this fine dissemination of pyrite, it is necessary to break down the coal in relatively small sizes in order to free such material. When this is done, it involves the washing of fine material, with associated troubles in regard to moisture and sludge recovery. To do the best possible work and utilize this coal in the best possible manner, we need the following:

1. Improvement in our methods of screening fine material.
2. Improvement in efficiency and economy of equipment for washing small sizes.
3. Effective methods of moisture removal without recourse to heat drying.
4. Improved methods of recovering and utilizing sludges.

Even if this were possible of accomplishment, we would still be confronted with the use of coals higher in sulphur than those to which the coking industry has been accustomed.

The difficulty in screening to small sizes has been due to moisture in the natural slacks. Possibly this would be overcome to a large extent if the mine-run were crushed to say minus ½-in. and we could do fairly efficient screening with present equipment.

If objectives 2 and 3 were accomplished, the screening might be improved by using wet screening ahead of the cleaning equipment. However, the discouraging part of the picture is the recovery and utilization of the sludges, which has been the great burden of the industry in Europe and in England for many years.

D. H. Davis and John Griffen (authors' reply).—Our purpose was to attempt to assemble as completely as possible all the data available on the subject; we realize that the future will present for solution more problems than have been solved in the past.

There seems to be general agreement that the acceptable sulphur limit for metallurgical coal will increase but that the economic advantage of low-sulphur coal in the manufacture of iron will bring to a greater usage some of the methods suggested for reducing the sulphur content in coal. A number of opinions were expressed in the discussion by men who have intimate knowledge of the effect of sulphur in coal on the operation of the blast furnace. Mr. King and Mr. Brown both advocate improvement of coal by all known means both underground and in surface plants, as the cost of improvement of the coal does not nearly approach the higher costs of the blast furnace when operating with poorer quality coals.

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[*] Engineering Editor, *Coal Age*.

[†] Jamison Coal and Coke Company.

It has been suggested by Mr. Eavenson and Dr. Powell that some of the sulphur will be removed after the iron is made. Mr. Eavenson points out that it has been shown that the added cost of washing coal for metallurgical purposes is definitely worth it and in recent years many cleaning plants have been built for this purpose.

Some persons responsible for the production and preparation of coal are often reluctant to undertake a program for thorough mechanical preparation and blending of coal. We believe that Dr. Powell's suggestion that the time is ripe for comparative economic studies of various expedients to reduce sulphur would do much to bring about a better understanding of all concerned on this important subject.

Mr. Zeller has clearly pointed out the principal difficulties encountered in coal preparation when fine crushing is adopted in order that a greater proportion of the sulphur may be removed by coal-cleaning methods. He has also listed the improvements needed in coal-cleaning technique in order to make it possible for the finely disseminated pyrite to be removed. In the past it has been found more economical to wash Pittsburgh-seam coal without preliminary crushing below 4-in. round hole. In most cases fine crushing has resulted in no improvement of the analyses of the washed coal unless oil flotation was employed. As there is no beneficiation below 100-mesh with launder, jig, or table methods of cleaning fine coal, the production of additional fines below 100-mesh will mitigate any improvement in the cleaning resulting from finer crushing, and for that reason it has not been the practice to crush under 4-in. in size.

We are in agreement with Dr. Young that bench or shelf would not be economically practical or beneficial to quality in mining the Pittsburgh seam. We can only point out that mechanical loading of the coal, so far, is producing a raw run-of-mine coal, which is more difficult and costly to clean to the same analysis than the run-of-mine formerly produced by hand loading. As our paper clearly shows, the bottom and top coals, which were largely unmined with hand loading, contain ash and sulphur-bearing impurities so intimately associated with the coal that a product of equal analysis cannot be obtained from them by known mechanical cleaning methods. Further, these parts of the seam generally vitiate the quality of the mechanically cleaned coal obtained from the rest of the seam. Mechanical loading includes a large proportion of the top and bottom coal in the mine output that formerly was not mined, and this is one of the principal causes for the increase in ash and sulphur in metallurgical coal.

It is natural that there should be a diversity of opinion in respect to the extent that selective mining should be practiced, and on the other hand to what extent the cleaning plant should attempt to reduce ash and sulphur. As the economic picture differs with each company, and especially since the raw-coal analyses of individual mines and companies show wide variation, we could not expect that everyone would agree as to the exact methods by which improvement in coals for the blast furnace are to be achieved.

Mr. Cassidy is of the opinion that selective mining is less justified in the Pittsburgh seam than in most seams and points out the advantages of mechanical loading en masse; that is, loading of coal and slate together in one loading operation. There is no doubt that this system has worked out very successfully, particularly in the various phases of mining. In regard to his statement to the effect that "quality and uniformity of clean coal can be and are now being realized at certain mines where selective mining has been discarded," we might say that quality and uniformity are capable of being measured and compared with either previous data on a different system in the same mine or with data on other mines, and that such statements would be more enlightening if supported by comparative records. His statement regarding quality and uniformity at these mines where mechanical loading en masse is used could be amplified by stating that some of the cleaning plants produced a high-gravity product in addition to the metallurgical coal. The variations in quality are mostly in this high-gravity product, which automatically improves the quality of the metallurgical coal. This high-gravity product has replaced steam coal formerly purchased on the open market and thereby makes it an economical proposition. The authors have advocated the practice of making a high-gravity steam coal, and where this is possible it will prove most beneficial in improving the quality of coal for metallurgical purposes. However, without the production of a steam coal, it is doubtful whether the cleaned coal from en masse loading would be suitable for metallurgical purposes.

We are thinking of a captive property where even the production of a metallurgical and steam coal by an efficient coal-cleaning plant would not meet the sulphur analyses being obtained in one of the mines referred to by Mr. Cassidy. Even if it were possible to reach the desired analyses in the metallurgical coal by the production of a secondary steam coal, the steel company can obtain its steam coal at a much cheaper delivered cost from another source. Even by selective mining and thorough cleaning of the coal, this mine cannot approach within 0.30 per cent the sulphur content of the metallurgical coal in one mine where en masse loading is practiced. Thus, we see that what may be right for one mine is not necessarily right for all mines in the district.

In order to thoroughly mix high-sulphur and low-sulphur coal, Mr. Cassidy suggests that elaborate arrangements of dumping cars from various sections are unnecessary, as the same or better results can be obtained by proper blending bins. We might ask: what are the particular features of these "proper blending bins" and what are the resultant uniform analyses? No doubt there is too much interference with the underground operations if it is necessary to hold cars at the main bottom until another trip of the desired analyses arrives. With sections of wide variation in sulphur content, it may be necessary to have a number of individual bins, each to contain coal of a certain range of sulphur. Blending of coal from the bins in proportion to the sulphur content of each would result in a mixture of more uniform sulphur analysis. If the variations in sulphur were wide, this could also be an expensive operation. It would, however, permit the mine to concentrate on mining, with its many problems, without being handicapped by an expensive and complicated dumping schedule.

Mr. Cassidy suggests that certain cleaning plants built to meet the requirements when mechanically loading the Pittsburgh seam en masse have successfully met the problems and are producing coal of acceptable quality and uniformity. We are fairly familiar with most of the cleaning plants built in the Pittsburgh district and have the impression that plants built with the intent of handling mechanically loaded coal have required at least an equal number of changes and additions to fit them to their duty as plants built before mechanical loading was in use, which subsequently were called upon to handle coal so mined. Our information on cleaning-plant performance indicates that in attempting to clean a mechanically loaded raw coal to the same standards of quality obtained from hand-loaded coal, both types of plants would fail equally to produce a coking coal of acceptable quality, because the inherent properties of the coal produced by mechanical loading could not under any system now in common use be cleaned to the desired point. From available information, this appears particularly true of many of the properties in the undeveloped area of Greene and Washington Counties, which will not be able to achieve even by properly designed preparation plants a sulphur analysis approaching present-day analyses. Therefore, either it will be necessary to raise the limits of acceptable sulphur, continue some of selective mining with highly efficient mechanical cleaning, produce by mechanical cleaning a steam coal of higher ash and sulphur so that a lower ash and sulphur metallurgical coal may be obtained, or to do all three.

To date, the developments in mechanical mining and benefication methods, although of definite economic advantage, cannot be considered a final and satisfactory answer to the use for cokemaking of the higher sulphur coals, which constitute such a large proportion of the reserves of the Pittsburgh seam in Pennsylvania. Mr. Eavenson intimates that a similar situation exists regarding the large reserves of this seam in northern West Virginia. We hope that a paper covering the reserves in West Virginia may be presented to the Institute during the next year or so. The statement by Dr. Powell, "In view of the growing sulphur problem in the Pittsburgh seam that the time is ripe for comparative economic studies of various expedients to meet the situation," is an excellent suggestion for future A.I.M.E. papers or a symposium.

Coal in Turkey

By Ferit Gurses,* Student Associate A.I.M.E.

(New York Meeting, February 1943)

EXTENSIVE coal and lignite deposits exist in Turkey. Bituminous coal is the nation's principal mineral resource; important not only as fuel for the industrial development of the country, but also as a means of obtaining foreign exchange. Of the electrical power generated during 1939, 82.8 per cent was derived from coal.[1] It is anticipated that the lignite deposits will be one of the major sources of power for the electrification of the country.

BITUMINOUS COAL

The locations of the four principal bituminous-coal[2] basins are indicated on Fig. 1. The Eregli-Zonguldak basin lies on the Black Sea northwest of Ankara; the Canakkale basin lies near the junction of the Dardannelles; the Trakya basin is in European Turkey northwest of the Bosphorus and the Erzurum basin lies far to the east.

The Eregli-Zonguldak basin is the most important of the four. This coal basin was discovered in 1829 but did not arouse interest until 1848; up to 1865, its annual production had not exceeded 50,000 metric tons. In 1865, during the Sultanate, working of coal mines was started by the Navy department, and the annual production was 61,145 metric tons. In 1908, the Minister of Commerce and Agriculture was authorized to develop the whole coal basin. Production reached 264,397 metric tons in 1910, and during 1915–1922, preceding the revolution, the total production of the basin was 2,676,043, or an average of 334,000 metric tons annually.

The period of national revolution that followed was marked by a steady increase in coal production in Turkey, although world production decreased, because of the crisis of 1929–1932, about 52 per cent.

AREA AND GEOLOGY OF THE COAL BASIN

The geology of the Eregli-Zonguldak area is not yet completely known. The basin begins at Gungazi, to the west of Eregli, and extends to Sogutozu, a distance of about 106 miles. The coal beds, however, appear only in places and the overlying Cretaceous strata hide them elsewhere. This area includes the most important centers, such as Kandilli, Alacaagzi, Teflenli, Kozlu, Gelik (see Fig. 2). The geological sequence of the district is correlated with the Cretaceous of the Balkans; which is, in turn, a prolongation of the Alpine zone.

Geological surveys made by G. Ralli, R. Zeiller and P. Arni[3] show that the Eregli-Zonguldak formations fall into three natural divisions. Series *a*, The Alacaagzi in the district of that name (Fig. 2), is of the Culm stage. The strata strike generally east-west and dip about 30° south. This series appears also around Kandilli and Camli, with a steeper angle of dip, but the deposits there lack regularity. The Alacaagzi stage appears also farther east, to the south of Kozlu and Gelik. The Alacaagzi series contains 15 seams of an average thickness of 2½ ft. of good coal.

Series *b*, the Kozlu, best represented in Zonguldak, Kilimli, Catalagzi, and between

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* Graduate student, School of Mines, Columbia University, N. Y.

[1] References are at the end of the paper.

Kozlu and Gelik, is of the Westphalian stage. In the first three places the strata dip almost vertically and outcrop along an east-west line of faulting. These vertical beds, five in number, called the Kilic series, constitute a transition zone between the Alacaagzi series itself and that occurring between Kozlu and Gelik. Southeast of Kandilli these seams are particularly good, and they are the principal source of production for the Kandilli mines. The Kilic seams also appear south of Kozlu and Gelik (Fig. 3). The series between Kozlu and Gelik is the best of the district, because of its regularity. About 20 veins can be distinguished, representing 130 ft. of coal.

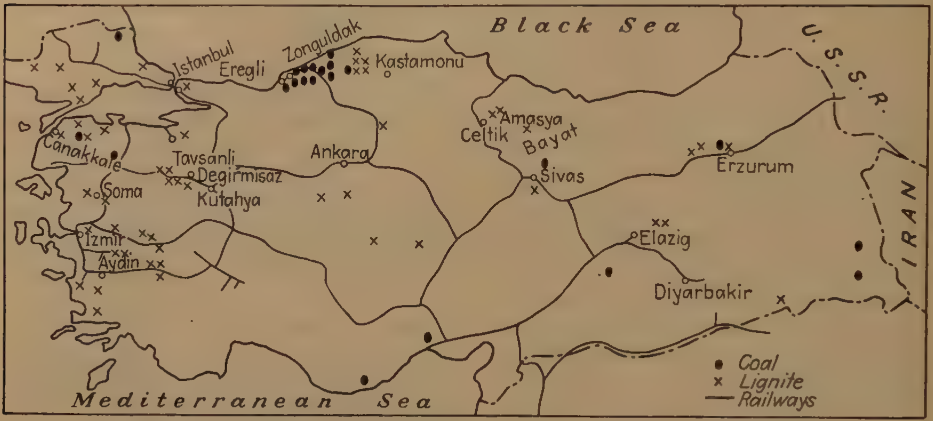

FIG. 1.—LOCATIONS OF BITUMINOUS COAL AND LIGNITE IN TURKEY.

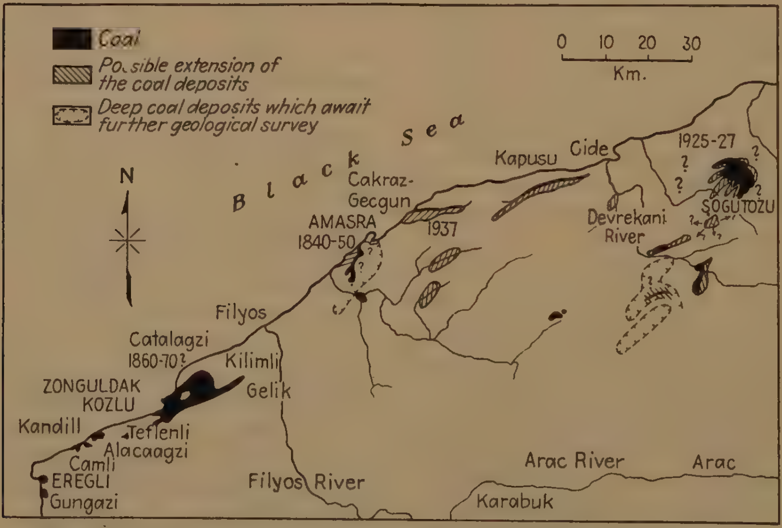

FIG. 2.—THE EREGLI-ZONGULDAK COAL DISTRICT. (*After W. S. Grancy.*)

The Karadon series (*c*) to the south of Kozlu and to the south and north of Gelik (Fig. 3), correspond to the upper division of the European beds, and seem to contain eight seams of poor coal. It occurs also at Amasra and Sogutozu, to the east of the Zonguldak basin, where it assumes a more interesting character.

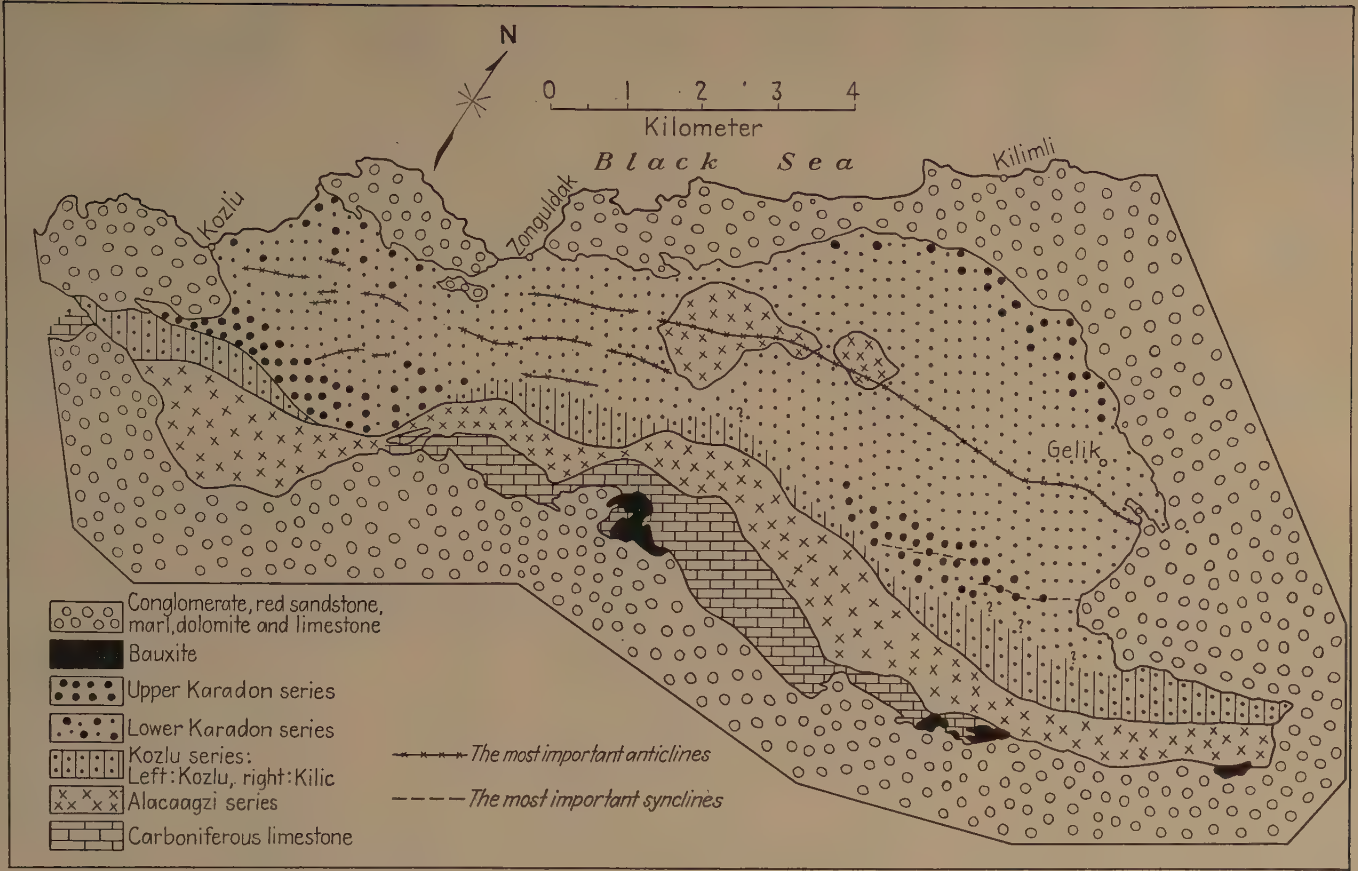
N
0 1 2 3 4
Kilometer
Black Sea
Kozlu
Zonguldak
Kilimli
Gelik
Conglomerate, red sandstone, marl, dolomite and limestone
Bauxite
Upper Karadon series
Lower Karadon series
Kozlu series: Left: Kozlu, right: Kilic
Alacaagzi series
Carboniferous limestone
The most important anticlines
The most important synclines

FIG. 3.—GEOLOGICAL MAP OF THE BITUMINOUS COAL BASIN OF ZONGULDAK. (*After P. Arni.*)

More careful geological study needs to be given to this coal district in order to permit estimation of its coal resources; even in the best known area of Zonguldak and Kozlu little is known above +980 ft. and below −650 ft. from sea level. It is hoped that the basin is rich in coal, which has been estimated to amount to approximately 40 million metric tons per square kilometer. If it is assumed that the reserve of the Eregli-Zonguldak area is 2 billion metric tons,[4] not more than 2.3 per cent of the reserve tonnage had been mined to 1940.

QUALITY OF COAL

The Eregli-Zonguldak coal is suitable for most purposes; it even compares favorably with the average of European bituminous coal. The percentage of volatile material varies between 30 and 40 per cent, when ash and moisture are deducted. It is difficult to give an analysis typical of the coal of the basin, since the calorific power and the amount of volatile matter of different coal beds, as well as the result of treatment for each, are variable. The following figures show only average limits:[2]

	PER CENT
Moisture	2–8
Ash	10–12
Volatile matter	26–32
Carbon	55–62
Coke	65–72
Sulphur	0.3–1
Calories	6,800–7,500

Coal from the lower stage has a higher percentage of volatile matter; therefore it is used in the manufacture of illuminating gas. Factories, railways, and steamship companies, on the other hand, utilize the coal from the middle stage.

DOMESTIC AND FOREIGN COMMERCE IN COAL

Turkish coal is sold commercially as large (+50-mm.), nut (50 to 18 mm.), hazel (18 to 10 mm.), slack (10 to 0 mm.), and *tout-venant* (run-of mine). The proportion of these size ranges remains almost constant, as follows: 14 per cent, large; 22 per cent, nut; 19 per cent, hazel; 45 per cent, slack.

These sizes are sold either separately or mixed, according to the demand. Factories prefer a definite size, whereas steamships or railways buy mixed coal. Mixed coal is sold in the following proportions: 30 per cent large, 35 per cent nut, 35 per cent hazel; and 20 per cent large, 25 per cent nut, 25 per cent hazel, 30 per cent slack.[2]

The coal is shipped from the ports of Zonguldak, Kilimi, Kozlu, Kandilli, Amasra, Kirecli, but Zonguldak is the only port that has modern loading equipment. Export coal moves mostly in foreign ships. Table 1 indicates the foreign markets to which Turkish coal was consigned during 1938.[5]

TABLE 1.—*Coal Shipped from Turkey in* 1938[5]

Destination	Percentage	Destination	Percentage
France	39.89	Rumania	2.97
Italy	20.21	England	2.91
Greece	17.18	Algeria	0.84
Brazil	8.91	Germany	0.64
Syria	5.97	Others	0.48

TABLE 2.—*Coal Shipped from Turkey between* 1928 *and* 1934[2]

Years	Amount, Metric Tons	Value, Turkish Pound[a]
1928	93,000	1,417,500
1929	155,500	1,876,000
1930	155,600	1,316,900
1931	215,800	1,661,000
1932	333,553	3,783,000
1933	479,360	3,820,700
1934	692,266	3,204,000

[a] A Turkish pound at present is worth approximately 76 cents in American money.

From 1927 to 1940 coal was produced by the following mines:[6] Eregli Komur Co., 37.07 per cent; Turkis and Kilimli Co., 18.55; Kozlu Komur Co., 13.83;

Turk Komur Kozlu Co., 9.12; Turk Komur Kandilli Co., 8.64; others, 1279.

The quantity exported between 1928 and 1934 and its value are shown Table 2.[2]

LIGNITES

The lignites of Turkey are mostly Tertiary in age. Their calorific values range between 4800 and 5600 for recent cormations; dense lignites, however, vary from 6000 to 7400 cal. There are two principal regions of lignite in Turkey, occidental Anadolu* and oriental Anadolu. The reserves of occidental Anadolu are rich and relatively clean.

TABLE 3.—*Analysis of Lignite*[7]

Analysis of Lignite as Received, Per Cent	
Water	43
Ash	15
Volatile matter	21.6
Fixed carbon	20.4
	100.00

Proximate Analysis of Pure Coal (Ash and Water Excluded), Per Cent	
Carbon	70.80
Hydrogen	4.70
Oxygen	19.54
Nitrogen	2.18
Sulphur	2.78
	100.00

Distillation Test of Lignite (Fisher Apparatus), Per Cent	
Water	45.80
Gas	8.00
Tar	3.10
Coke and ashes	43.10
	100.00

The most important lignite beds are in Soma (Fig. 1). They are of an excellent quality, having a calorific value of 5400. Besides these, there are the Degirmisaz and Tavsanli (Kutahya) beds, which constitute an important reserve.

A notable characteristic of this basin is that as the strata extend toward the south the overburden increases progressively. At the north it is comparatively thin, so that in certain regions the coal is almost at the surface, and parts of this deposit can be mined as open pits.

The analysis made upon more than 350 samples of Seyidomer lignites shows the results[7] given in Table 3.

This lignite is not a good combustible, because of its high percentage of water and ash. The main economic value of this poor fuel is for the local production of electrical energy. The large quantity of lignite (about 90,000,000 metric tons), the regularity of the deposit, and the good ground (requiring less timber for underground working), all contribute to this.

The lignite deposits of oriental Anadolu (Fig. 1) are rather irregularly scattered; the most important ones are near Bayat and Celtik. A geological survey made by Stchepinsky in 1939 shows lignite deposits occurring at Sivas, but those are low-grade because of the unfavorable climate of the Oligocene period, during which the lignite was formed.[5] These resources are not yet fully developed, but the Celtik-Amasya basin produced 34,995 metric tons during the first 8 months of 1940.

Table 4 shows lignite production in the most important districts of Turkey during first 8 months of 1940.[6]

TABLE 4.—*Production of Lignite in Turkey from January to August,* 1940[6]

METRIC TONS

Month	Soma	Degirmisaz	Tvsanli	Celtik-Amasya
January	4,671	1,255	4,099	4,844
February	6,845	3,294	3,941	5,046
March	7,550	4,585	4,741	5,155
April	5,413	2,392	2,744	4,448
May	7,426	3,022	1,880	4,466
June	6,042	3,260	1,800	4,289
July	5,434	2,178	3,255	3,422
August	7,383	3,258	3,015	3,305
Total	50,761	23,240	25,505	34,995

Comparing these results with those of 1939, Soma shows an increase of 74.57 per cent; Degirmisaz 46.64 per cent; Tavsanli 68.33 per cent; and Celtik-Amasya 30.27 per cent.

Besides providing a raw material for the making of several by-products, increased production of lignite would cause an improvement in the national economy, for the following reasons:

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* Name given to Asia Minor.

1. More bituminous coal could be exported, instead of being kept for domestic consumption, which would increase foreign exchange.

2. The substitution of lignite for *tezek* (animal excrement dried and used as combustible during the winter) would make possible the use of *tezek* as fertilizer.

3. Lignite can be used for the production of electrical energy.

CONSUMPTION AND PRODUCTION FACTORS

One of the major economic problems of modern Turkey is its coal industry. It is a source of export revenue, and also has special importance in the industrial development of the country because petroleum and water power are not used. Since the beginning of economic organization in Turkey, there has been a nationalistic effort to insulate the domestic economy from fluctuations of world markets, and to make it more nearly self-sufficient. The motives behind this action, from the economic point of view, are obvious: to proceed rapidly toward industrialization and raise the standard of living of the country.

This objective prepared the way for organization of the coal industry under government control in an attempt to secure larger and more efficient production. This resulted, finally (No. 3851 Decrie 2/1289, effective April 12, 1940), in bringing all coal producers under state supervision. The amount of capital necessary to open new fields was thereby made available through the government, and centralized financial control in this industry offers, at least theoretically, some prospects of successful production.

In its relation to the government, the coal industry has two characteristics that make it different from the other mineral industries of Turkey. In the first place, it enjoys the unusual privilege that laborers in the coal districts are required to work in the coal mines, regardless of their inclination, in order to meet seasonal labor shortages. Secondly, the coal industry is controlled to keep coal prices at an approved level. Thus they have been kept constant in Turkey during periods of fluctuation elsewhere.

This has come about because the Government believes that the economic activity of the country depends on an increased coal output and maintenance of a reasonable domestic price level. Governmental control in the coal industry was a natural consequence of previous inadequate management and unsatisfactory production, under the present economic system of Turkey. It would be expected that such an important industry could not escape some form of control if it was not functioning competently. To forecast the trend of this control is difficult in an age of transition and uncertainty like that of today; nevertheless, it seems probable that such a system, presumably rigid and arbitrary, would experience difficulty in presenting an immediate solution to the coal problem.

The present study, therefore, is concerned primarily with factors involving coal consumption and production.

The coal market in Turkey has been steadily expanding, primarily because of increased consumption per capita due to higher standards of living, and also because of increased demand for coal by newly constructed plants.

The upward tendency in coal consumption is due partly to the growth in population through birth and immigration. As long as population increases, the demand for coal will increase, but the growth of consumption is much faster than that of population. Between 1933 and 1940, the rate of coal output increased three times as fast as the rate of population. The increased consumption per capita between 1931 and 1940 was as follows:

Years	Pounds per Capita	Years	Pounds per Capita
1931	111	1936	132
1932	114	1937	160
1933	115	1938	170
1934	129	1939	207
1935	130	1940	262[1]

Table 5 shows the increase in consumption made by public utilities and industries between 1933 and 1940.[1]

TABLE 5.—*Increase in Consumption by Public Utilities and Industries*

Consumer	Coal Consumption, Metric Tons		Increase in Coal Consumption, Per Cent
	1938	1940	
Maritime Services			65.7
Cement Mfg. Co.	18,581	61,506	33.3
Railways	178,960	459,196	25.7
Electrical energy			20.9
Gas manufacturing			18[1]
Sugar industry			12[1]

[1] Approximately.

Paralleling the expansion of the coal market itself, increased ease of access to new markets has resulted from development of the railroads. This factor, as far as can be foreseen, is the one that probably will play the greatest part in continued expansion of coal consumption.

TABLE 6.—*Output in Eregli-Zonguldak District*

Year	Output, Metric Tons	Net Tons per Man/Day	Explosives, Kg.	Timbers, Cu. M.	Power, Kw-hr.
1923	597,499	0.551	14,381	18,522	350,880
1924	994,020	0.550	23,926	30,815	439,943
1925	1,957,625	0.550	23,050	30,644	471,602
1926	1,216,008	0.606	29,269	38,912	795,256
1927	1,323,833	0.508	31,865	43,686	1,034,920
1928	1,250,639	0.606	45,148	43,772	1,743,800
1929	1,421,008	0.605	51,129	52,577	5,359,370
1930	1,595,359	0.661	57,599	62,219	9,149,822
1934	2,288,269	0.677	101,890	78,308	19,007,094
1935	2,340,491	0.700	111,813	78,630	19,549,825
1937	2,306,869	0.577	132,791	106,907	28,288,194
1939	2,696,397	0.534	160,462	129,284	38,455,552

Table 6 gives data on the Eregli-Zonguldak district between 1923 and 1939. The totals for the 17 years were 28,895,400 metric tons; explosives used 1,243,925 kg.; timber, 1,079,921 cu. m.; power, 223,210,267 kilowatt-hours.

Table 6 indicates that the output per man-day has not changed appreciably. After rising irregularly from 0.551 net tons per man-day in 1923 to 0.700 in 1935, it dropped to 0.534 in 1939; the average being 0.612 between 1923 and 1939. The increased annual output therefore is secured only by additional labor, not by increased efficiency of production. The man-day coal output in Pennsylvania is about seven times greater than that of Turkey, and that of England is about twice as great. That this is due to physical conditions does not seem likely, because Turkish coal miners are not handicapped by difficult underground conditions; they are not forced to work at depths as great as 3500 ft., nor to mine seams as thin as 14 in., as are British miners. The low man-day output seems due, mainly, to the lack of use of modern equipment or its less effective use, to management, to the irregular service of the mine laborers and their attitude toward their jobs, and to an inadequate supply of timber.

The mechanization of Turkish coal mines is one of the country's major future problems. It will require better coordination of production factors and development of a supply of permanent mine labor. Under present conditions increased use of modern equipment is not likely. The longwall method does not require the use of loading equipment, since most of the working faces are sufficiently inclined to slide the broken coal down to the chutes. The responsible authorities are taking every measure possible to secure better use of present mechanical equipment; lectures are given to the miners and a new school of mines has been opened, with training in a four-year curriculum.

Since mining machinery is not yet manufactured in Turkey, the mechanization of Turkish coal and lignite mines, as well as those of chromite, copper, iron, lead and manganese, will have to depend on imported machinery and associated equipment. A more liberal use of machinery may be expected when labor costs tend to be higher and power becomes cheaper.

Although the main haulageways are equipped with Diesel locomotives, the performance of the haulage systems is a source of trouble. Its inefficient operation limits not only the capacity of the mines but also requires additional manpower. The figures in Table 7, computed from the performance data[8] of one mine in 1940, reflect this situation. The haulage efficiency of the second column represents the amounts of coal hauled per shift divided by the number of men employed in repairing, timber carrying, shoveling, hauling, unloading; that is, all the operations not included under the first column. According to these figures, the average number of men required (except for the face) to maintain 1000 net tons hourly capacity is about 10,550 in this particular mine. Furthermore, the number of men required, in all coal mines, to maintain 1000 net tons hourly capacity (reciprocal of man-hour) was 11,420 in 1935. This is about seven times greater than that of American bituminous coal mines for the same period. These data indicate that machines and men are not doing their best work; that the haulage is not functioning properly; and that, presumably, some absenteeism has its effects.

TABLE 7.—*Performance of One Mine in 1940*[8]

NET TONS

FACE EFFICIENCY, TONS PER MAN	HAULAGE EFFICIENCY, TONS PER SHIFT PER MAN
1.943	0.713
1.640	0.651
1.056	0.693
0.200	0.115
2.680	0.645
1.910	0.899
2.740	1.136
3.100	0.833
0.977	0.880
2.297	1.020

Management reflects the effects of a controlled economic system. Owing to the absence of normal competitive forces, its activity is not governed by the corresponding economic factors. The main objective of the coal industry is its contribution to the national welfare. However, meeting this national need by producing more and securing corresponding results seems to lack momentum, at least for the moment, because of the lack of incentives within the structure of the organization. From the production point of view, it may be claimed that the present form of management and production system is satisfactory, provided it is assumed that the planning agencies under state control possess the desire, the will, and ability to attain eventually reasonably efficient production. Only more years of experience will provide the evidence to support this assumption.

The labor situation also deserves consideration. The number of laborers employed in the coal districts increases from year to year. Some of them intend to become miners and establish themselves in the district; some want only to earn money to pay their taxes, and then return to their farms; and some of them are there only because they are required by law to be there. When laborers leave the mines for their farms during harvest time a difficult labor situation is caused, and the recruiting of laborers causes even more difficulty. Men who are recruited and required to work for a certain period are mostly local agricultural labor; they are a sensitive and proud people who resent compulsion and present a corresponding attitude toward their work. It is not usually known whether there will be enough labor available for the next season or even next month, since this depends on weather and harvest conditions. During the summer of 1936, for example, the crops were exceptionally good, and the coal mines lacked laborers during that period; in certain localities the deficiency of miners had reached 50 per cent of its normal level.[9]

There are several reasons for this fluctuation in supply and the general lack of men skilled in coal mining:

1. The men are not paid enough; in 1937 the wage of a miner was of the order of one Turkish lira, which was about 80 cents. Although the standard of living is low, this wage level is still not normal. But in spite of the low wages of the miners, the labor cost does not seem to be low, because of the difficulty and expense of recruiting.

2. Housing conditions in the coal district are not very satisfactory. Fortunately the Government is helping to improve the living conditions a great deal; more than half of the 21,361 laborers in Zonguldak now live in new barracks.

3. The present day's pay system. It seems perfectly possible to introduce the contract system for some classes of mine labor. The coal deposits are not generally irregular and short-term contracts may avoid uncertainties. Underground conditions are not so difficult as to make a low output inevitable. The eventual result of this would be that fewer men would be needed for a given output and no compulsory method would be necessary to provide labor supply.

As long as the labor situation remains as it is it will not be possible to hope for reasonably high productivity and improved efficiency. Eventual mechanization of the mines by modern equipment would not prove helpful unless a permanent supply of skilled miners were made available.

Lack of timber supply for underground support presents another difficulty in obtaining better results in the coal industry. Turkey does not produce much steel and none for underground use, therefore only timber is used. Although the country has great forest resources, the poor quality of the timber, lack of transportation facilities and irregularity of supply make the problem of underground support difficult. Heavy ground conditions in the district require an excess use of timber, which amounts to 1.54 cu. ft. per net tons of coal mined, and call for additional manpower for its handling and transportation.

SUMMARY AND CONCLUSION

Extensive deposits of bituminous coal and lignite occur in Turkey. Coal has an important role in the industrial development of the country.

The average annual coal production from 1902 to 1921 was 626,000 net tons; it rose to 1,870,000 net tons between 1923 and 1939.

Owned by the State, the coal mines are operated under government supervision. Unlike other Turkish industries, labor shortages are met by recruiting men under a draft law, and the output of various mines is marketed through a single sales agency at a controlled price.

Labor productivity is still low, its average being 0.612 net tons per man-day between 1923 and 1939. This situation is caused mainly by the inadequacy of labor supply and the lack of experienced miners, resulting from low wages, poor housing conditions, and lack of incentives for production.

More general adoption of modern equipment and improvement in transportation facilities, and an adequate and regular supply of underground supports, coupled with a more effective organization in the management, seem necessary to make coal more cheaply available in the nation's markets. Only through such means can Turkey promote its national welfare, a possibility that is realizable from the good quality and large reserves of its coal resources.

In spite of the difficulties encountered, the coal industry of Turkey is showing progress in economic activity; Zonguldak, in particular, resembles an American boom town.

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DISCUSSION

M. M. LEIGHTON,* Urbana, Ill.—From the standpoint of quality of the coal resources, it is evident that the Eregli-Zonguldak field is the most important body of coal in Turkey. This coal is of middle Pennsylvanian age, similar to that of most of the bituminous coal of Carboniferous age mined in North America. It appears to be of high-volatile A and medium volatile rank. The average quantity of the coal is estimated to be about 100 million tons per square mile. The total quantity is approximately half that originally making up the Herrin (No. 6) coal bed in Franklin County, Illinois.

The author expresses dissatisfaction with the quantity of coal produced from this field. This dissatisfaction rests largely upon the low rate of per capita output. It is interesting to note that mining is essentially nonmechanical—that is, hand mining is used; and, what is more significant, that compensation is on the day's pay and not the tonnage system. It would appear that such a system of compensation would be conducive to a low rate of man-day output. The Oriental contract system commonly characterizing labor employment in the Far East is only day's pay system once removed, but in general seems to work better. The fact that the labor is to a considerable extent compulsory would also probably tend to lower the per capita output.

The development of more industrial enterprises and elevation of the standards of living would appear to be the best bases for expansion of mining. But because of the relatively small supply of this relatively high-grade fuel, it seems probable that every effort should be made to substitute lignite for bituminous coal wherever this can be done. The relatively small reserve of bituminous coal seems to point to the desirability of careful conservation of the coal for internal use, thereby extending as far as possible the life of this important energy resource for use by the Turkish people rather than its use to obtain foreign exchange.

The adoption of mechanized mining would considerably reduce the cost of production, probably, but might lead to undesirable economic and social repercussions unless it paralleled increased demand. The boom conditions described as existing in the Zonguldak area are an indication that increased demand is under way.

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Breaking Coal with Cardox in the Pittsburgh Seam

By E. C. Skinner*

(New York Meeting, February 1944)

Cardox, which consists essentially of a steel tube containing carbon dioxide compressed to the liquid state, is a trade name designating a device used principally in coal mines to break down coal.

The use of Cardox to break down coal was developed in the Middle Western coal fields around 1927 or 1928. At first, a large, heavy tube was used, requiring many parts. Through experimentation and technologic improvement, the weight of the tube has been brought down from 88 lb. to 20 lb. and the new, light tube makes use of larger volumes of CO_2 at lower pressures than formerly was thought possible.

This blasting method is extensively used today in coal mines and probably one tenth of the bituminous coal produced in the United States is prepared with Cardox.

Construction and Application

As used in the coal mine, Cardox looks like a piece of steel pipe 2 to 3 in. in diameter and 3 to 5 ft. long. When it is necessary to fire a shot, the tube is wired with two leg wires that are long enough so that when the tube is pushed the desired distance into the hole the wires will protrude 5 to 6 in. It has been found that No. 18 insulated copper annunciator wire is the best material for leg wires. A two-conductor rubber-covered shot-firing cable of sufficient length is then attached to the leg wires. A 2 to 6-volt current is applied, either by means of shot-firing batteries or the standard type of shot-firing generator.

Upon the application of the current, the charge in the cartridge is set off. Because the breaking action upon the coal is supplied by the relatively slow physical expansion of the carbon dioxide gas rather than by the rapid chemical formation of gases as in more familiar explosives, the result is a slow heaving action, somewhat similar to that produced by black powder, rather than a sharp detonating effect. Permissible powder does its work at roughly ten times the pressures in use with Cardox.

The escaped gas from a Cardox shot apparently works along the planes of weakness in the coal face, such as the coal cleats, the edges of the cut, and along the top of the seam, breaking off the coal toward the free faces and heaving the mass of coal outward.

Cardox tubes are manufactured in various sizes to take care of variable conditions of seam size, undercut depth, cleatage, coal hardness, and so forth.

The tubes range in diameter from 2 to 3 in. (Table 1) and the normal CO_2 charge is from 24 to 96 oz. For any one tube size, several sizes of shear disks are available, thus excellent control of the type of shooting desired is possible.

In general, local experience and experimentation dictate the size of tube chosen for any particular job. For high coal, and deeper or wider cuts, a larger size is chosen to start with, and the opposite is true for low coal, softer coal, or small cuts. After the tube size has been experimentally selected, tests are run in the mine to determine how many shot holes are needed and where the holes should be placed. The size of the disk,

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* Pittsburgh Coal Co., Library, Pa.

which determines the pressure at which the CO_2 is released, may then be varied to give harder or softer shooting, whichever is required. Often, by experimenting with various sizes of disks, it is possible to eliminate one hole to a face; or it may also be possible, by drilling an extra hole, to use a thinner disk, thus producing much coarser coal.

Choice of tube, disk size, and placement of holes are, of course, directly related to the loading of the coal. With machine loading, it is very important that the coal be shot well enough so that it can be loaded efficiently without digging coal from the face. On the other hand, if the coal market requires a minimum of fine sizes, it is equally important that the coal be prepared, shot, and subsequently handled in a manner designed to satisfy this requirement. Thus for commercial coal mines shipping coal into a market where coarse sizes command a premium price, a delicate balance must be set up between the shooting and the loading, in order that the most economical procedure be followed. In such a case, the goal to strive for is to shoot the coal in a manner to give maximum coarseness yet hard enough to give the loading machine a chance to operate efficiently. This balance is best obtained by experimentation at each mine.

TABLE 1.—*Permissible Combinations of Cardox Cartridges and Other Pertinent Data*

1	2	3	4	5	6	7	8	9	10	11	12	13
Tubes Size	Shear Disk, In.	Disk Gauge	Disk Thickness, In.	Hydraulic Pressure to Shear Disk			Recommended Charcoal Heater, Grams	Maximum Heater Permissible, Grams	Recommended Normal Charge CO_2, Oz.	Minimum Heater for Shear with Normal CO_2, Grams	Minimum CO_2 for Shear with Heater in Column 8, oz.	Over-all Assembled Length, In.
				Minimum by Bur. of Mines	Normal	Maximum, with Cardox						
2–50	1 1/16	16	0.062	8,000	10,000	12,000	40	80	24	25		34 3/4
		14	0.078	10,000	12,000	14,000	45	80	24	30	16	
		12	0.109	13,000	15,000	17,500	60	100	24	35	16	
		10	0.140	17,000	19,000	22,000	100	110	24	50		
2–80	1 1/16	16	0.062	8,000	10,000	12,000	60	80	36	35	23	46 3/4
		14	0.078	10,000	12,000	14,000	80	110	36	40	22	
		12	0.109	13,000	15,000	17,500	100	110	36	60	24	
		10	0.140	17,000	19,000	22,000	110	110	36	80	26	
2–100	1 1/16	16	0.062	8,000	10,000	12,000	60	110	48	40	31	58 3/4
		14	0.078	10,000	12,000	14,000	80	110	48	45	27	
		12	0.109	13,000	15,000	17,500	100	110	48	60	38	
		10	0.140	17,000	19,000	22,000	110	110	48	80	39	
231–130	1 5/8	12	0.109	12,000	14,000	16,000	100	140	64	50	50	59 5/8
		10	0.140	16,000	18,000	20,000	140	160	64	70	46	
		8	0.172	18,000	20,000	22,000	180	180	64	100		
3–200	1 3/4	12	0.109	10,000	12,000	13,000	180	220	96	90	64	53
		10	0.140	13,000	15,000	17,000	200	220	96	110	73	
		8	0.172	17,000	18,000	20,000	220	220	96	160	77	

Fig. 2 shows a diagrammatic sketch of a Cardox tube. The tube body is forged of chrome-molybdenum alloy steel 3/16 in. thick for this particular size. This is the chamber that contains the CO_2. The top assembly is fastened to the shell body by means of heavy threads, with a gasket between the tube body and the top assembly. Contained in the top assembly are the electrical connections, both of which are connected by means of leg wires to the shot-firing cable. The electrode, a part of the top assembly, but insulated from it, provides the electrical connection between the terminals and the heater element after the heater element has been positioned, and the top assembly screwed down. The heater element consists of a paper tube about ¾ in. in diameter and 14 in. long over all, fitted with a wooden plug top constructed with a shoulder, so that it rests in the throat of the tube body and is held firmly in place when the top assembly is put on. Two enameled wires extend about ⅓ down into the heater element, entirely surrounded by a mixture of potassium perchlorate and charcoal. One wire comes up out of the center of the wooden plug and makes contact with the positive electrode. The other wire is brought out through the side of the wooden plug at the shoulder and makes contact with the tube body forming the ground when the tube is assembled. A fine nichrome wire coated with fulminate of mercury joins the ends of these two wires together within the heater element.

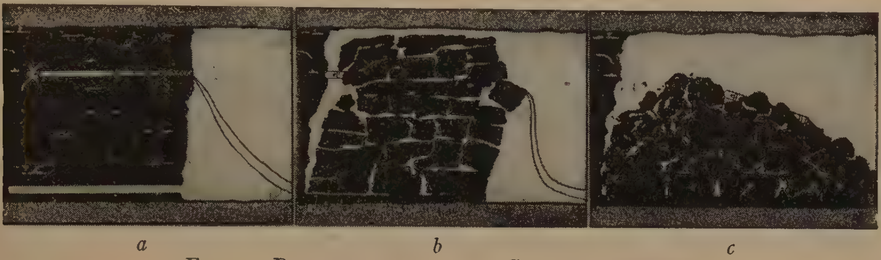

FIG. 1.—BREAKING ACTION OF A CARDOX DISCHARGE.

a. Tube in place at instant of disk rupture. *b.* Discharge expended, coal breaking down. *c.* Coal shot down.

Also contained in the top assembly is the set-screw admission valve, used when refilling the tube for admitting the CO_2 into the tube body.

At the lower end of the tube, or the end placed at the back of the shot hole, is the discharge cap, also screwed into the tube body. Held between the tube body and the discharge cap is a gasket and the shear disk.

When a proper electrical impulse is applied to the circuit of the tube, the mixture of potassium perchlorate and charcoal is ignited by the fulminate of mercury. The heat generated by the rapid burning of the material in the heater element gasifies the carbon dioxide contained within the shell body and greatly increases its volume until the pressure becomes great enough to rupture the shear disk, allowing the gas to escape with a surge. These shear disks vary in thickness from 0.062 to 0.172 in. (Table 1), depending upon the size of shell, and are made of common soft steel. The thickness of the disk determines when it will rupture and thus controls the pressure at which the surge of CO_2 escapes from the tube.

Angle vent ports are placed in the discharge cap, to reverse the direction of some of the escaping gas, thus creating a tendency to hold the shell in the shot hole.

RECHARGING

After discharge, the tubes are recovered and sent outside the mine for recharging. The tube is first dismantled by taking off the discharge cap and the top assembly, and at this time the parts are inspected. The burnt heater element and any remaining portions of the sheared disk from the previous discharge are removed. The tube is then reassembled by the insertion of a new heater element and shear disk, as well as new gaskets at each end of the tube body. The discharge cap and top assembly are then screwed tightly into place and the tube is ready for refilling. It is placed in a fixture with the top assembly clamped tightly in the filling mechanism of the high-pressure pump, which pumps the liquid CO_2 from the insulated storage tank into the tube. When the tube has been charged to the proper pressure, it is disengaged from the pump and weighed, in order to check the amount of CO_2 in the tube. The electrical circuit within the top assembly is checked with a galvanometer and finally each end is dipped in water, to be sure that there are no leaks at the gaskets. This completes the charging operation and the tube is now ready for re-use.

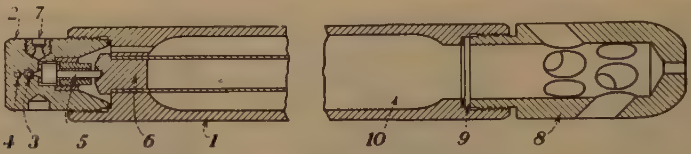

Fig. 2.—Section through an assembled Cardox tube.

1. Shell body.
2. Top assembly.
3. Positive terminal.
4. Negative or ground terminal.
5. Electrode.
6. Heater element.
7. Set screw, admission valve.
8. Discharge cap.
9. Shear disk.
10. Carbon dioxide reservoir.

The CO_2 may be stored at the plant in 4-ton storage tanks, which can be replenished periodically by an insulated tank truck that hauls the CO_2 from a main distributing point in the district.

Heater elements, because of their inflammability, are stored in fireproof buildings.

Characteristics of Pittsburgh Seam

Several characteristic features of the seam occur throughout all areas.* Typical are the "bearing-in bands," consisting of two bands of slate from ½ to 1 in. thick, separated by 3 to 4 in. of coal. These bands present a problem in underground face preparation, in that it is advantageous, from the standpoint of free impurities in the coal, to keep the bands in large pieces and not to have them broken up. In hand loading, it is the practice to gob them in the mine, but with mechanical loading they are loaded out with the coal. The shooting should minimize breakage at these strata.

Another regular feature throughout the area is the draw slate, which occurs immediately over the coal. The characteristics of the draw slate itself, however, are extremely variable. It ranges from 0 to 5 ft. in thickness, from white to black in color, and from extremely hard to muddy in texture. Upon exposure to the mine air, it draws away from the roof immediately above and falls and crumbles. The speed with which this drawing takes place is dependent to a large extent on the hardness of the slate. Where the draw slate is hard, very little contamination of the coal takes place because of drawing and falling, but where it is soft, or wet, it begins to crumble as soon as the coal is shot. From a quality standpoint, it is desirable to keep the slate out of the coal, and here again the shooting practices should be set up to disturb the slate with the coal shots as little as possible.

Other natural irregularities that occur in the seam are rolls of bottom or top rock, pinching out part or all of the coal, and clay veins and spare, which are fissures of varying thickness through the seam filled with shaly or clayey material. These problems must be solved individually as they are encountered.

The coal, from 6 in. above the bottom of the seam to the bottom bearing-in band, is of good quality but lacks solid structure. The coal from the top band to the draw slate is the portion of the seam which yields the coarser sizes. Due regard in the preparation and handling of coal from this section of the seam will result in the production of the maximum yield of coarse, blocky coal. The top breast also contains the best coal in the seam from the standpoint of ash, sulphur, and burning characteristics.

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* For details regarding the seam, see paper by Davis and Griffen, page 22, this volume.

There are two definite and distinct cleating planes in the coal—one called the butt cleat, which is the shorter of the two, and the other called the face cleat, which is long and very well defined. The two cleats are at approximately 90° to each other, which causes the mined coal to have a blocky appearance.

OPERATING ADVANTAGES

The Pittsburgh Coal Co. uses Cardox for shooting the coal wherever mechanical loading is done. Normally, the finer sizes sell under the average or mine-run selling price while the coarser sizes sell above the average. Manifestly the dictates of economics is to produce a maximum of coarse coal.

There are other operating advantages, which deserve serious consideration. Since mechanical cleaning usually accompanies mechanical loading, it is an advantage to clean the larger sizes rather than the smaller ones. Water cleaning methods of fine coal pose problems of water clarification and settling, and drying of the cleaned coal—all of which adds a burden to cleaning expenses.

According to data appearing in Chapter I of "The Economics of Coal Preparation," by J. B. Morrow and D. H. Davis, the following relationships hold true: Assuming the ratio of the cost of wet cleaning the ⅜ to 4-in. size as 1, it has been estimated that the cost ratio for hand-picking plus 4-in. is 1.15 and the ratio for wet cleaning the minus ⅜-in. size is 2.43. This is almost 2½ times the cost of cleaning the intermediate sizes.

Size deterioration may occur at any place along the line from the time the coal face is shot in the mine until it is put in the customer's bin. Coal shot with Cardox withstands handling, loading, dumping, screening, and storage, to a marked degree. This is attributed to the pushing, breaking effect, using lowest possible pressures, of the Cardox shot upon the original coal face. In a given piece of coal, there are fewer shatter cracks caused by the initial blast to weaken with each subsequent jar and finally to break into smaller pieces.

Because of the emphasis being placed on dust suppression in the mines, there is an advantage in handling coal that releases minimum dust. Extreme fines, as found in the immediate area of the coal face around a powder shot, are considerably reduced, which in addition to the lessened degradation into finer sizes during loading, transferring, or dumping reduces the amount of dust available to be thrown into suspension in the mine air.

The results of screen analysis of coal shot with Cardox as compared with coal shot with powder are shown in Table 2. This coal was loaded with mobile loaders into mine cars and the sizes were those at the final loading point after the coal had been put through the cleaning process.

The percentage of finer sizes, under 1 in., is reduced with Cardox by a total of 8 per cent, or about the amount the plus 4-in. size is increased. The intermediate sizes remain approximately the same. In addition, the quality of the plus 4-in. size is improved because there is more plus 6-in. included.

TABLE 2.—*Comparative Mine-run Size Consist of Coal Shot with Powder and Cardox at One Mine in Western Pennsylvania*

Size, In.	Size, Per Cent	
	Powder Shot	Cardox Shot
+6	9	17
4 to 6	9	10
2 to 4	18	19
1 to 2	20	18
−1	44	36
Total	100	100

In areas where coarser coal brings higher prices, this difference in size consist may have an important bearing on the total realization. The amount of increased realization that could be obtained also depends upon the proportion of the coal sold as the primary grades shown in Table 2 in relation to the amount sold as a mixture of grades or mine run. Any increase in realization, however, is partially offset by the increased cost of using Cardox.

Table 2 shows that the top sizes in the plus 4-in. size are improved. This is also true of the other sizes. Accordingly, a coarser slack is obtained, which, even though it may bring no higher realization than powder-shot slack, nevertheless is a more marketable product and normally sells more easily.

Slack from powder shooting, however, tends to be of higher quality analytically than Cardox slack because of its benificiation by the greater degradation of the "top-breast" coal.

In the Pittsburgh Coal Company mines, Cardox is used to blast the coal for mobile loading, hand loading on conveyors, and duckbill loading.

In most cases, the coal is loaded directly to the mine car, hauled to the outside, and dumped. At some mines, the coal is then mechanically cleaned at this location; at others, it is dumped to railroad cars for transportation to a central cleaning plant. Where there are installations of shuttle cars, additional dumping is necessary.

The coal is all worked on the room-and-pillar system, under cover varying in thickness from 100 to 500 ft. Generally main face entries are turned off main butts in sets of from 4 to 6, at intervals of about 2000 ft. Butt or room entries, usually in pairs, are turned off the main faces at intervals designed to give a 300-ft. room. Entries are driven 11 or 12 ft. wide on 40 or 50-ft. centers; rooms are driven about 21 ft. wide on 50-ft. centers, leaving a rib about 29 ft. thick to bring out on the retreat. The room entries are worked out retreating; that is, entries are first driven up and room necks turned. Rooms are then started on the inby end of the entry and a breakline is established. The rooms and ribs are worked out successively to the bottom or outby end of the butt entry. Room entries are driven on the butt course, so that the rooms, which are turned off at 90°, may be driven up on the face cleat. This provides a regular, straight, working face and leads to better coal preparation and a greater amount of coarse coal.

In general, in the mechanical mines, there are three basic types of work—entry or narrow work, single-wing room work, and rib cutover work—for which definite face-preparation standards have been set up. Within the usual range of working conditions, these practices give uniformly good loadability and production of coarser sizes. The fundamental rules of good face preparation pertain to all types of work.

Because a Cardox tube is a fixed charge and cannot be changed readily to take care of substandard preparation, places should be as similar as possible in width, undercutting, hole placement, and so forth.

Toward that end, cutting should be carefully done so as to leave straight ribs and an even, straight face devoid of pockets. Cuttings should be completely removed from the kerf, so as to leave a space in which to shoot the cut of coal.

Because of the presence of the bearing-in bands, it is the practice to drill snub holes. These holes are started above the bands and drilled at such an angle that the back of the hole lands at the back of the undercut and just touches the top band. The action of the snub shot is that of a breaker shot. It is drilled on a pitch and is always shot first, so as to kick the bottom coal out to provide a free face and leave space in which to shoot the upper section of the seam.

Holes drilled near the top of the seam are called top holes. Usually they are placed on each side of the cut and if the place is more than 12 ft. wide a hole in the center is added. These holes are drilled from 6 to 8 in. away from the slate, in order to minimize action on the slate. Top holes on the sides next to the ribs are kept 6 in. from the solid rib. All top holes should be drilled level and, except in rib cuts, parallel to the sight line of the place. In order to place holes at these exact locations, it is the practice of the drillers to use rods of iron of ½-in. diameter and 8 to 10 ft. long to poke into the kerf and sound out the outlines of the cuts. A rod is lined up under the cut where the hole is wanted and the hole is then drilled following the rod.

Cardox tubes, when placed in the top holes, are pushed completely to the back of the cut, then pulled out 4 in. in order to form a "cushion," and also to give some space for the tube to move in when it is discharged. Tubes in the snub holes are placed so that the top of the tube is just inside the collar of the hole. Because of the angle of the hole, the force of the discharge will travel to the back of the cut and heave the coal out, but there will be no tendency to overly shatter the bands.

Shot holes in both the coal and slate are drilled with post-mounted electric drills. The drilling pattern is so arranged that as many holes as possible may be drilled from one setup of the drill. Slate holes are drilled therefore at the same time as the coal holes. The drills are equipped with augers about 4 ft. long, fitted with a splined shank on one end and a socket to take an identical spline on the other. The cutter or auger head, in which the detachable bits are mounted by means of small wedge blocks, are also fitted with splined shanks. A hole is started with one section of auger fitted to the thread bar. After the first auger has drilled to its depth, the second is added and the hole is finished to the full depth.

When the drilling and other preparatory work on the coal face has been completed, the coal is ready to be shot. The usual practice is to shoot the complete round of coal holes at this time, but on special occasion this is not done. If the slate is exceptionally bad and likely to fall, or if the place is in a pillar undergoing pressure, it is common practice to shoot only the snub holes at this time. The shot-firer wires up the tubes as explained previously, places the shells in the holes, and pushes them into position with his stick (Fig. 3). The holes are then fired in correct order one by one. After the last shot has been fired, the empty tubes are collected and returned to the storage rack.

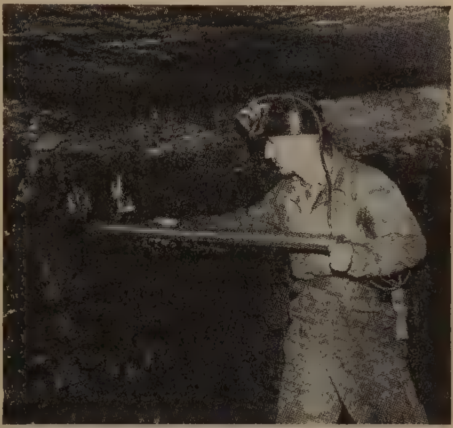

Fig. 3.—Shot-firer charging shot hole with Cardox.

Fig. 4 shows a coal face that has been shot with Cardox.

Face Preparation

Fig. 5 is a diagram of a single-wing room as it is worked in mechanical loading. The "wing" in this case refers to the block of coal on the right, measuring approximately 7 by 7 ft. This block or "wing" is kept 7 ft. behind the advancing face, so as to make it unnecessary to handle the slate over the full width of the room. The slate is taken down in the entire area of the advance face, but it is left intact and supported on posts in the "wing" area. Slate from the advance place is gobbed, where possible, in the mined "wing" area.

The round of holes in this place, consisting of four coal holes and three slate holes, is drilled from three drill setups. Hole No. 1 is in the wing and is drilled normally from a setup under the posted slate. When conditions do not permit this, the hole is drilled from an extra setup in the center of the place. The hole is drilled from a position under the slate, so as to throw all the coal in the wing out toward the center of the place, where it can readily be picked up by the loading machine. This particular hole produces the coarsest coal in the cut because of the three free faces to which the discharge is working.

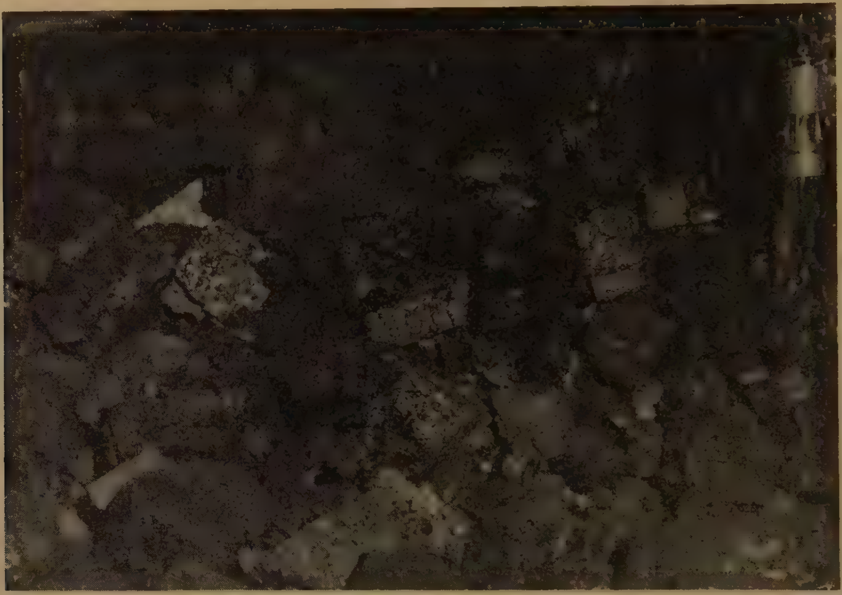

Fig. 4.—Coal face shot with Cardox.

The second hole to be fired is the snub hole in the advance face place. The shot breaks out a cone-shaped section of the bottom coal and cracks the bottom coal from the right rib to about two thirds across the face. The right-hand top hole is fired next, bringing down the coal on the top right side of the room. The final shot is in the left-hand top hole, which brings down the remainder of the coal.

Fig. 6 is a cut of coal in a rib cutover. It is the most difficult cut to shoot because of the possibility of weight. It is impossible at times to get all the cuttings from the kerf. It is also made more difficult because of the angle cut on both sides. The left side is angled in order to allow the loading machine access to the place. The right side is cut in this manner because of the necessity of removing the back stump to allow the overburden to cave readily upon completion of the cutover. This cut is shot with six holes in all, two of them being snub shots. The snub holes are drilled in the same relative position as the snub shot in single wing. Two holes are used in order to be sure that the bottom coal is kicked out before the top shots are fired. After the snubs have been fired, the top center shot is fired in order to break out a wedge in the center of the place. When the center has been cleared, the sides are shot into this open space, the left side being shot first, for reasons of safety. Hole No. 6 is shot with powder, because of its short length and to prevent the loss of a Cardox tube in the gob. All of these holes are drilled from two drill setups.

In entries or narrow places, as shown in Fig. 7, only three shots are required to prepare the cut for loading. The first hole fired is the snub drilled in the usual manner. The right top hole is fired second, after which the left top hole is fired to complete the shooting. This small cut requires, however, two drill setups.

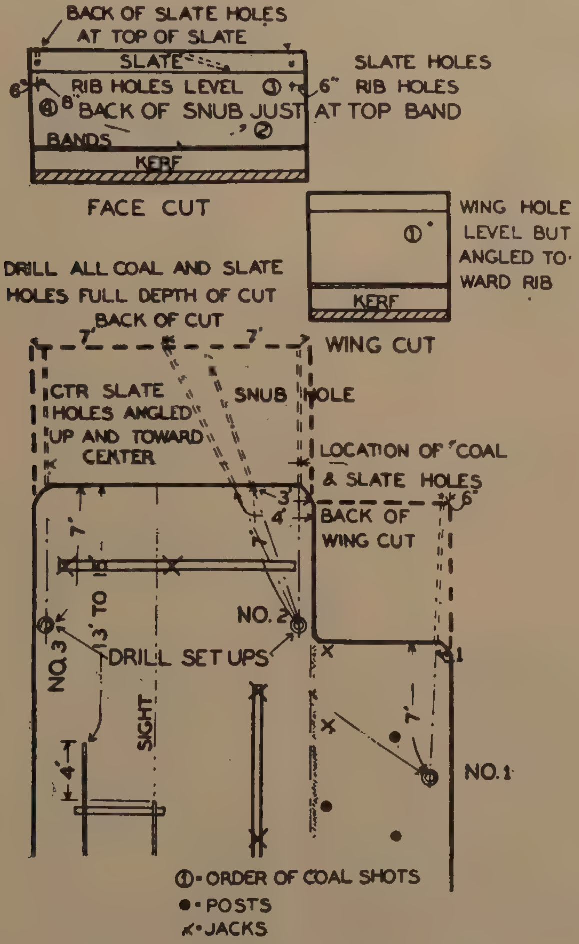

FIG. 5.—DIAGRAM OF A SINGLE-WING ROOM.

All of the foregoing schemes for face preparation are used in order to get the best possible loading conditions along with maximum amount of coarse coal. By the use of Cardox in combination with definite face-preparation standards, it is possible to get a degree of control in shooting and coarse size production not possible otherwise. Where the system is correctly applied, there is enough shooting power available to bring down the coal if drilling, cutting, and cleaning of the kerf have been properly done. The Cardox tube is a standard shot designed to do a certain amount of work within certain limits, and it is necessary therefore that all companion operations be up to standard. If any of the other preparation practices are neglected, the coal is not likely to be well prepared for loading. With powder shooting, preparation mistakes can be covered up by overshooting, but with Cardox this cannot be done.

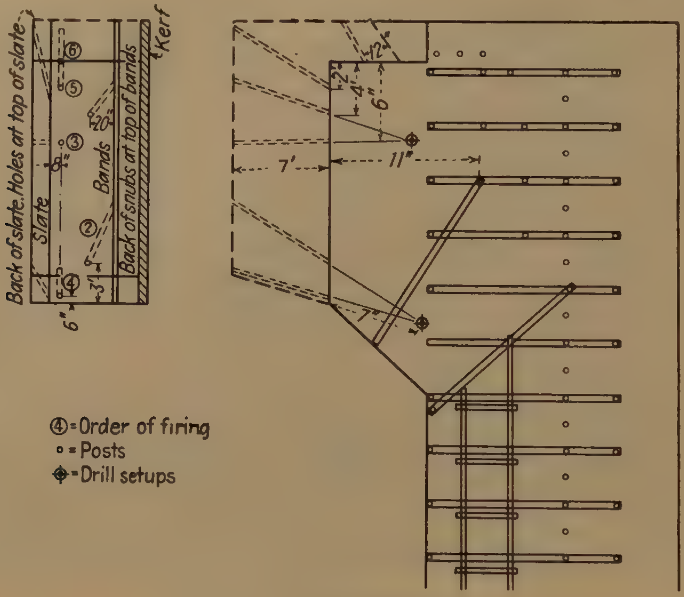

FIG. 6.—RIB CUTOVER CUT.

This inability to cover up is an advantage rather than a disadvantage. Because mistakes most often show up as tight coal and consequent delay to the loader, it focuses the attention of the supervisors upon the need for getting each job done correctly. It definitely discourages the use of poor work habits. As the work is the same in all cuts of the same kind, the operations are more easily taught to inexperienced men, and the whole cycle tends to become routine and standardized.

The decision as to the best way to shoot a cut of coal no longer rests on the judgment of the individual charging the shot hole, but rather on the basis of all the facts of experience and experimentation applied from an engineering point of view.

There are other factors that may be considered in the use of this device for dislodging coal. One of these is the absence of smoke and fumes. When the operating cycle demands it, the loading machine may start immediately after a cut has been shot down, with no discomfort to the men. Visibility is practically the same as though no explosive had been fired, and the only odor present is the slight smell caused by the burning of the contents of the heater element.

The transfer of loaded and empty tubes between the charging station and the working place is easily accomplished. The interchange usually is made once a day, but at large mines it may be desirable to interchange twice, in order to reduce the number of tubes in service. A quota of tubes is set up for each loading machine and they are then traded evenly, a load for an empty. Each loader section is fitted with a rack upon which all tubes not in the coal are placed at the end of each shift. The Cardox delivery man visits each station and replenishes the supply, taking the empties back to the charging plant. The tubes are hauled in insulated wooden cars.

Perpetual inventories of each section are kept as required by the Federal Explosives Act, but even without Government requirement this would be done because losses may be spotted more quickly than in any other way. Wastage of tubes, due to loss or damage beyond repair, amounts to about 2 per cent per month.

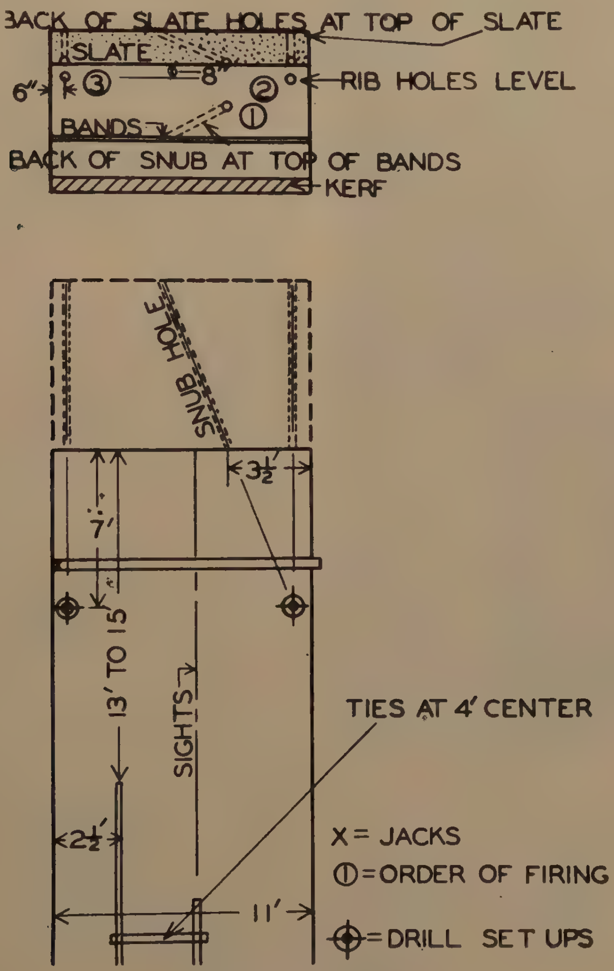

Fig. 7.—An entry cut.

Cardox Tubes as Fire Extinguishers

Carbon dioxide gas, which is contained in each shell, is an excellent fire extinguisher and a system has been set up to utilize this characteristic in emergencies. Placards are posted at each Cardox rack in the mine, explaining in detail how the CO_2 within the tube may be released to extinguish fires. This is done by loosening the set-screw admission valve by the use of a 9⁄16-in. Allen wrench, several of which are attached to the instruction placard. These wrenches are also carried on each gathering locomotive. After the set screw is opened sufficiently to allow the gas to escape, the shell may be thrown on the fire or the CO_2 jet may be played on the blaze in the same manner as a stream of water. The CO_2 blankets the atmosphere in the immediate vicinity of the fire to such an extent that combustion is no longer supported, and the fire is extinguished.

FIG. 8.—RESULT OF A FLYING CARDOX TUBE.

In hand-loading mines where Cardox is not used for breaking coal, stations containing a half dozen tubes have been set up at strategic locations, such as repair shops and pump houses, to be used in an emergency for fighting small fires.

In numerous instances, tubes have been used for this purpose with good results. At one mine, the Joy operator neglected to remove the power tap of the trailing cable from the trolley wire during lunch period. A defective splice in a length of cable piled on the Joy head blew through and ignited the cable insulation, creating a small but hot blaze. When the fire was discovered, the tap was removed and a number of Cardox tubes were opened and tossed onto the pile of burning cable. Fortunately the fire had not yet ignited the coal, so that it was quickly brought under control.

SAFETY

Cardox has been classed by the U. S. Bureau of Mines as a permissible blasting device when applied in conformance with the regulations developed for its use. By experimentation and testing, permissibility has been granted on the safe combination of heater ingredient, weight of CO_2 and disk thickness.

Its outstanding safety feature is the complete absence of flame upon discharge. In the discharge reaction of the shell, no part of the burning heater element is exposed to mine atmosphere because it is encased in a gastight cylinder surrounded by inert gas. According to Bureau of Mines *Report of Investigations* 3714, "all Bureau of Mines laboratory tests have failed to obtain an ignition of any mixture in air, of gas and or coal dust by Cardox devices."

Cardox, however, has been responsible for a peculiar type of accident, as the tube is ejected with great force from the shot hole upon discharge when proper care is not exercised. When this occurs, the tube may travel only to the collar of the hole and fall out or it may be propelled with such force as to travel several hundred feet with great velocity (Fig. 8).

Causes for tube ejection consist principally of the following: (1) placing too much burden or work on a single tube, (2) attempting to discharge a tube in a hole that is on the solid, (3) use of a defective tube, (4) failure to "cushion" shots.

Accidents from flying tubes, while not numerous, are exceptionally serious and they continue to happen in spite of the precautions taken. During the years 1941, 1942, and 1943, there were 613 compensable accidents in Pittsburgh Coal Company mines, of which six were the result of flying Cardox tubes. One of the six was a fatality. Four of the accidents occurred in

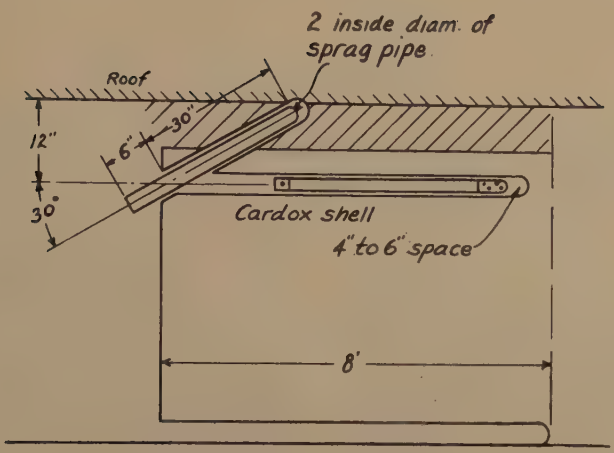

FIG. 9.—PIPE HOLDING DEVICE.

1941. Company rules and regulations governing all shot-firing forbid shooting from holes drilled in the solid. The rules call for the use of a shot-firing cable of sufficient length so that the shot may be fired from a safe place—in addition to other safeguards. Investigations reveal that the tubes are ejected because of failure to heed the precautions.

In conjunction with the Bureau of Mines, an effort was made to find a method or device to keep tubes in the hole under all conditions at the face. After much experimentation, an effective device was developed, which if used at all times, practically eliminates the hazards from ejections. It consists merely of a short angled hole in any direction (Fig. 9) into which is placed a piece of strong pipe about 3 ft. long, which blocks the shot hole at the collar. Since the installation of this scheme, no cases of tubes ejected from any hole so protected have been recorded.

ACKNOWLEDGMENTS

The author wishes to thank Mr. J. E. Norton, Assistant to the President, and Mr. D. H. Davis, Product Control Manager, of the Pittsburgh Coal Company, and Mr. Curran Cavanagh, District Manager of the Cardox Corporation, for their help in preparation of this paper.

DISCUSSION

(H. F. Hebley, presiding)

H. P. GREENWALD,* Pittsburgh, Pa.—It may be well to call attention to the fact that the Bureau of Mines' requirements for the shooting of Cardox were changed quite recently—in fact, within the past two or three weeks—to eliminate the old requirement of stemming.

MEMBER.—Is it conceivable that CO_2 could be lost through the filler valve? Would it fire?

H. P. GREENWALD.—There have been defective gaskets, but if the leak is not discovered at the time of filling it is quite minute and the flame would be ejected through a very small orifice. I am advised that the question of defective gaskets was considered in gallery testing and that ignitions were not obtained if a permissible size and type of heater was used. Nonpermissible heaters did give ignitions.

The report published by the Royal Commission of Mines in Great Britain in 1938 discussed Cardox somewhat favorably. Of course, at that time British experience was limited, but at least there had not been an accident. The report on the White Haven colliery disaster a number of years ago discussed the possibility of an ejected shell driven through a steel tub being the cause of an ignition of gas. This may appear far fetched but it was cited as the possible cause of a bad explosion.

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[*] U. S. Bureau of Mines.

Stripping Pitching Beds in Pennsylvania's Anthracite Region

By D. C. Helms,* C. E. Brown† and O. W. Shimer,‡ Members A.I.M.E.

(New York Meeting, February 1943)

The early history and progress of anthracite stripping, from the first known operation at Summit Hill in 1821 through 1917, was covered in 1917 in a paper by J. B. Warriner,[1] then chief engineer, now president of the Lehigh Navigation Coal Co., and stripping methods between 1917 and 1931 were the subject of a paper in 1931 by H. H. Otto,[2] of the Hudson Coal Company.

The present paper is confined to methods used today on pitching beds in the western middle and southern anthracite fields.

The anthracite industry, not unlike other large industries, had been affected for more than a decade by the adverse economic conditions of the country. High development and operating costs and the competition of other fuels caused some collieries to be closed temporarily, while others were permanently abandoned.

In order to obtain a larger tonnage at a lower development cost, a larger stripping program was put into effect by many companies.

In 1932, stripping operations accounted for 3½ million tons, or 7 per cent of the total anthracite production of the year, and in the next 10 years this increased until more than double that amount, or over 14 per cent of the total production, was from stripping operations. These figures represent the total production of the entire anthracite region; unfortunately, the figures have not been separated from the different fields. There are some operations in the southern and middle western fields where the tonnage obtained from stripping amounts to more than 30 per cent of the total colliery production.

Stripping operations on pitching beds present difficult problems. Owing to the various geological formations, a different situation is encountered in nearly every stripping pit; and because of the proximity of an overlying bed, it is very often necessary to load the entire overburden into trucks and haul it out of the pit to the dump area, although back filling and casting is done whenever possible. Beds in the areas described in this paper pitch from 20° to 90° and some are on the invert in many sharp folds and faults.

Planning

The more successful strippings have had adequate engineering analysis. Some work is being done without much preliminary planning, but many strippings would have failed completely if close supervision by the engineering department had not been maintained.

Locations of stripping areas are determined as follows:

1. By constructing cross sections over the areas and drilling sufficient diamond-drill or well drill holes to study the strata and to determine the location, thickness and pitch of the beds.

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[1] J. B. Warriner: Anthracite Stripping. *Trans.* A.I.M.E. (1917) **57**, 159.

[2] H. H. Otto: Stripping in the Anthracite Region. *Trans.* A.I.M.E. (1931) **94**, 181.

2. By projecting the pitch of the bed on a rough cross section from points at which its location is known, in areas previously mined, to points where it should intersect the surface, and starting a stripping shovel at one of those points.

3. By making a cut across an area with a dragline or stripping shovel at right angles to the strike of the bed formation.

4. By noting the location of the outcrop by examining bootleg operations or test pits.

The first method is by far the best. By assembling all data from the cross sections and boreholes, the thickness and quality of the bed may be noted, also the character and amount of overburden, especially the thickness and hardness of the rock, and the ratio of overburden to coal may be established. The cross sections also show the adjacent beds, and a determination is made of the amount of overburden that must be hauled out or may be side-cast without covering up other outcrops. The latter is important, as this overburden would later have to be removed at a considerable expense if it were decided to strip the adjacent beds. With this method better cost records are kept on the operation by running cross sections over the area stripped each month, to determine whether or not the stripping is being prosecuted to the greatest economical depth.

In the use of the second method, there have been times when a stripping shovel has been started at the point of the assumed intersection of the bed and the surface, only to find that the bed was on the inverted dip, or the angle of the pitch changed before reaching the surface and the bed was missed entirely.

Many strippings have been started by the third method, which proved profitable to the contractor. At other places, however, considerable sums of money have been spent in ditching across outcrops and no strippable coal has been found.

Profitable strippings have been made by the fourth method, but, like the second and third methods, it has the disadvantage of not offering any definite way of determining the economical depth of the stripping and whether or not that depth is being maintained. Many thousands of tons of coal have been recovered from areas that have been bootlegged, or are in isolated sections, because the contractor has been willing to take a chance. It is more or less a hit or miss proposition, and the contracts are generally on an unclassified basis, little or no planning being done in advance.

Needless to say, the more successful contractors insist on working from definite plans, so that there may be close cooperation between the contractor and the engineering department.

DRILLING

Considerable progress has been made in the past few years in drilling and blasting of the rock overburden. There is a definite trend toward changing from 6-in. to 9-in. drill holes for blasting, with either electric or Diesel-driven drilling equipment.

In some strippings where extremely hard sandstone and conglomerate are encountered, some 40 to 50 ft. thick, the success of the operations has been determined by the use of larger drills. Because of the increased weight of the stem of the 9-in. drill, it is possible to drill a given depth from 30 to 40 per cent faster than with a 6-in. drill, which permits the spacing of holes farther apart, so that 30 to 40 per cent more rock is shattered for the same number or length of holes.

With the introduction of the 9-in. drills, most contractors began to deck-load the holes; that is, to concentrate the charge of powder in the hardest rock, or at the place of the greatest resistance, in order to create a maximum degree of fragmentation. A careful record of each hole is kept as it is drilled, so that the location of the hard rock is known and the blasting charges are placed accordingly. The blast holes usually are charged with 40, or 50 or 60 per cent dynamite, depending on the kind and the position of the rock. Gelatin dynamite is used in wet holes.

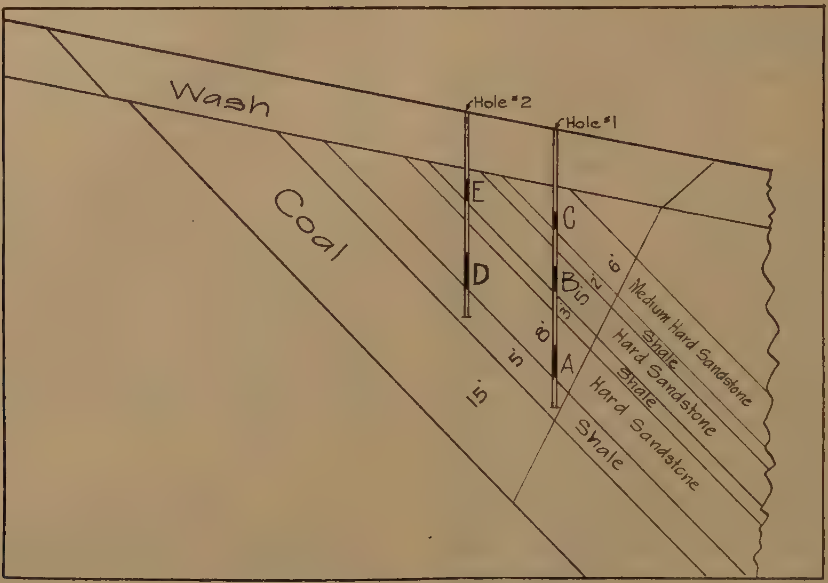

FIG. 1.—DECK LOADING IN BLAST HOLES.

Fig. 1 shows a typical section of the strata and the method used in placing the blasting charge. In hole No. 1, 150 lb. of dynamite is placed near the bottom of the hard sandstone *A*, 100 lb. at *B*, which is also in the hard sandstone, and 50 lb. at *C*, which is medium hard sandstone. In hole No. 2, 150 lb. of dynamite is placed at *D* and 75 lb. at *E*. In strippings covered in this paper, from $\frac{6}{10}$ to $\frac{8}{10}$ lb. of dynamite is used per cubic yard of rock blasted. Fine culm, clay, or drill cuttings from the blast holes are used for stemming and to separate the different charges in each hole.

In the vicinity of villages, 6000 lb. of dynamite is the usual blasting charge, and in isolated areas blasting charges of 30,000 lb. are not uncommon.

Primacord is used almost exclusively to detonate the charge. As all blast holes are connected to one main line, only one blasting cap is necessary to explode the entire charge.

EXCAVATING

The type of stripping equipment employed in areas of steep pitching beds depends on the degree of pitch and the surface conditions encountered. In pits where the bed is on a pitch of 90°, the overburden is comparatively light, consisting of clay and gravel, and space for the spoil material can be found on either side of the pit within reach of a small dragline. The coal is loaded with the same dragline, which usually is a machine with a 2 to 4-cu. yd. bucket and a 60 to 80-ft. boom. This is especially true where the thickness of the vein allows just enough clearance for the passage of the bucket of a small machine. The depth to which the coal is removed depends on the reach of the dragline, and often just how long the rock side walls stand up before they break and slide into the pit.

In other pits, where the pitch of the bed varies from 55° up to 90°, and hard overlying rock is encountered, larger draglines, with 5 to 9-yd. buckets and booms up to 200 ft. are used. Generally these are the Monighan, or walking type, either Diesel or electrically powered. As the pitch decreases, and it is possible to operate a shovel down in the pit, the overburden is removed with draglines and the coal is loaded with power shovels.

Where the economic limit or depth of the stripping has been reached, there is always a certain amount of coal below the mechanical limit of the shovel that has been used to load the coal, which must be recovered with a dragline (Fig. 2). The truck or haulage road is necessarily on top of the coal and it is the coal in the triangle *ABC* that must be recovered with a small dragline. This is accomplished by starting on the inside end of the pit and retreating outward. The greater the angle or pitch of the bed, the deeper will be the cut in the coal. As the pitch increases, the width of the cut, or V, will decrease, making it necessary to raise the boom to an angle that will allow ample clearance for the swing of the bucket.

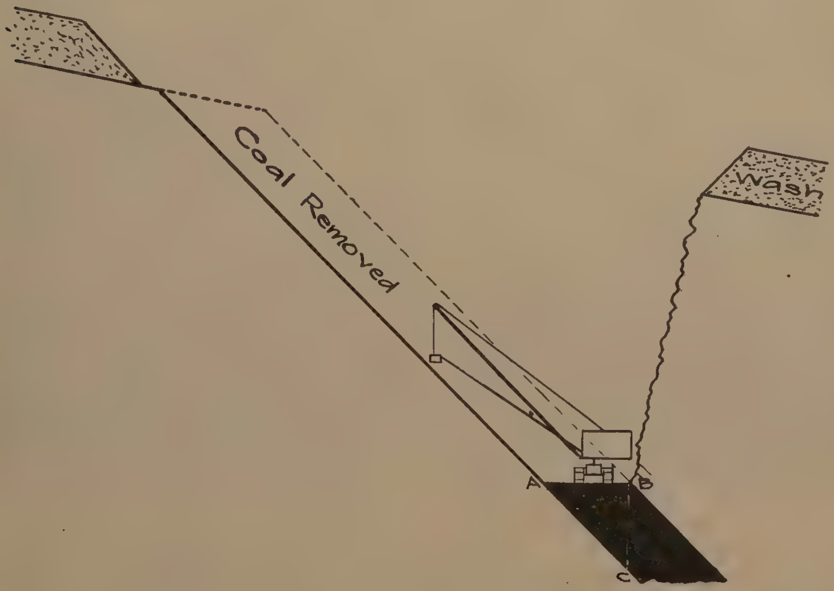

FIG. 2.—POSITION OF DRAGLINE IN NARROW CUT.

A great many 2 to 4-yd. draglines are in use and still have their place in the anthracite region in areas where the coal is on a heavy pitch. Because of the ease of mobility of this comparatively small equipment, there are many small areas of coal in isolated sections, near the top or on the side of steep sloping mountains, which otherwise could not be recovered economically.

In strippings where the pitch is not too great and most of the overburden must be hauled out by trucks, close-coupled power shovels are used in loading. The largest shovels of this type are equipped with 6-yd. dippers. Where it is necessary to load the trucks at the top of the cut, loading must be done with a dragline.

La Tourneau carryall wagons have also been introduced in areas partly covered with ash or slate banks, and where the wash is thick enough to justify their use.

TRANSPORTATION

Motor-truck haulage is now generally used; however, there are a few exceptions, where rail haulage is used for transportation directly from the stripping pit to the refuse dump, cleaning plant or breaker. The largest trucks now in service are 12-cu.-yd. end-dump Euclids, and 9 and 11-cu.-yd. Stirling and Mack trucks, and in one location 21-cu.-yd. Mack trucks were recently added. The majority of these are equipped with Diesel motors.

No trailer or semitrailer trucks are used in any of the strippings covered in this paper, as the narrow pits limit the space in which a truck might be turned. The trend is to increase the capacity of the present type of truck, which is equipped with dual rear tires. In line with this idea, tire manufacturers are cooperating by increasing the durability of the tires. To increase the size of the truck body by using sideboards, or by other means, without giving due consideration to the size and construction of the tires leads to increased tire expense.

The use of motor trucks permits more efficient and flexible pit operation and has led to increased shovel performance. With a fleet of trucks in operation the trucks may be spotted on either side of the shovel; this cuts down the arc of the swing in which the shovel must move in loading, and also eliminates the time that would be necessary for changing and spotting trips of railroad cars. Grades up to 15 per cent are now used and a great deal of the back-switching to attain rail grade out of the deep pits has been eliminated. Maximum grades on railroads in stripping pits where steam locomotives are still being used are usually 3 per cent; in a few instances, 5 per cent for short distances.

In some cases the coal loaded into trucks at the stripping is hauled either to a ramp, where it is transferred to railroad cars, or direct to the breaker, but in most instances it is hauled to a rough cleaning plant where the larger sizes of refuse are removed.

Not all of the areas are in virgin coal; many of the strippings are in areas that have been first-mined and caved—or partly robbed; others are in areas that have been partly mined by bootleggers. Where these conditions exist the coal is mixed more-or-less with the overlying refuse, and, while the use of smaller shovels and draglines eliminates much of the refuse in the loading operation, it is found more economical and efficient to send the coal to a rough-cleaning plant, which is as near the stripping as possible. In some areas, coal from two or more pits is sent to the same cleaning plant, which is as nearly centrally located as possible. In these cleaning plants, all of the material is passed over a bull-shaker, where the minus 6-in. material is eliminated and by-passed directly to the railroad car. The plus 6-in. material is passed over a picking table, where all large pieces of rock and refuse are removed and dropped into a rock pocket and later hauled by truck to a refuse dump. The material that goes over the end of the picking table is passed through a set of large rolls before it reaches the railroad car. While the bullshaker and the picking table are driven by the same camshaft, they are so arranged that they oscillate counter to one another, and in this manner keep the vibration of the equipment and structure to a minimum. There are of course some modifications to this type of cleaning plant but the type described above is in general use.

Most contractors now use bulldozers and motor graders to make truck roads and to maintain them in good operating condition, and in conjunction with this equipment road sprinklers are used to lay the dust and to prevent the surface of the roads from drying and crumbling into dust under usage by the large trucks.

Virtually every stripping of any consequence between Nesquehoning and Shamokin, a distance of 40 miles, has been visited by the authors. Strippings in which different kinds of equipment are utilized and those in which occur many of the problems on pitching beds have been selected as representative of the conditions in general and are hereinafter more fully described.

Philadelphia and Reading Coal and Iron Co.

Monitor Stripping—A-30

The Monitor stripping on the Holmes' Four Foot and Mammoth beds is at Locust Summit, 3 miles southwest of Mt. Carmel, in the Mahanoy Basin. It is 600 ft. wide and 4500 ft. long. The Dick Construction Co. has the contract for removing the overburden.

Before work was started, the area was cross-sectioned and the overburden was tested with well-drill holes. Estimates of quantities were made, to determine the depth or limit to which work might be extended.

Very close inspection is given by the engineers to this stripping operation, in order to maintain continuity in the removal of rock and coal. Points are set for each blast hole and for the benches of rock, and grade stakes are set, to make certain that the truck roads are kept as closely as possible on a grade of 8 per cent. Every shot or cut is measured as soon as it is cleaned out.

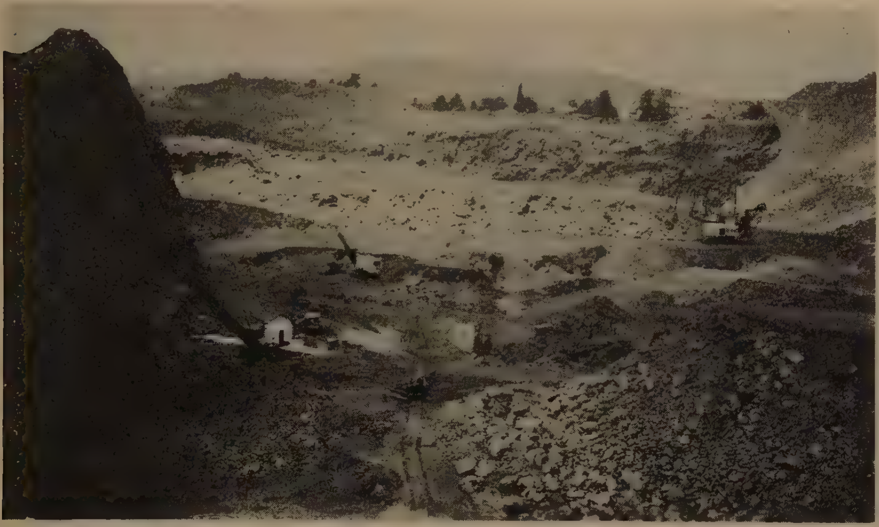

Fig. 3.—Monitor stripping.

On the north side of the stripping, the bed pitches 55° to the south to a short syncline, or roll, and after rising a short distance above the local basin, the coal continues on a south dip of 22°. Fig. 3 shows a general view of this stripping. On the extreme upper right can be seen the pitch of the bottom rock, above which the Mammoth bed has been removed. The bed on the left side of the picture is on a pitch of 22°. The coal from three separate beds is being recovered in this stripping pit, which takes in an area 600 ft. wide and 1400 ft. long.

It is customary for the Philadelphia and Reading Coal and Iron Co. to construct cross sections over the entire distance of what might be considered as strippable areas, and then confine a contract to a given distance or area until it is satisfied that all the economically strippable coal has been recovered in that area.

The overburden consists of 6 ft. of clay and gravel and 8 ft. of soft sandstone over the first, or Holmes, bed, which averages 6 ft. in thickness. Five feet of slate separates the Holmes from the Four Foot bed.

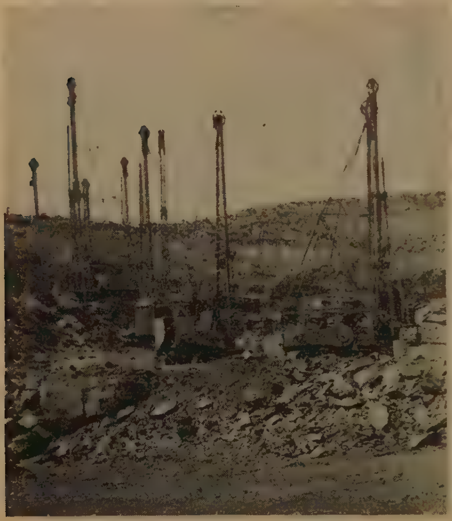

Fig. 4.—Monitor stripping.

The Mammoth bed, which lies 20 to 50 ft. below the Four Foot bed, is 30 ft. thick, the rock interval is very hard sandstone and conglomerate. A roll extends east and west across the whole area.

Two No. 170 Bucyrus-Erie electric shovels, each equipped with a 40-ft. boom and a 6-cu.-yd. dipper, are used to load the rock in the area over the Mammoth bed. These shovels are the extra heavy-duty type, similar to those used on the Iron Ranges, and are the only 6-cu.-yd. shovels of this type in the anthracite region. The shovels are operated three 7-hr. shifts, or 21 hr. a day, five days a week and each loads an average of 115,000 cu. yd. of rock per month. On a test run one of these shovels loaded twelve hundred 12-ton trucks in 21 hours.

An additional No. 120-B Bucyrus-Erie electric power shovel, with a 38-ft. boom and a 5-yd. dipper, is used to load the overburden that lies over the Holmes bed. This shovel, working on the same basis as the other two, loads an average of 80,000 cu. yd. per month, or a combined three-shovel monthly output of 310,000 cu. yards.

Forty 10 and 12-cu.-yd. Diesel-driven Euclid rear-dump trucks haul the rock 3500 ft. to an abandoned stripping pit that is being filled in; two 21-cu.-yd. Mack trucks were added to the fleet in August 1942. The trucks are end dumpers with 10 wheels, with two pairs of dual wheels in tandem at the rear of the truck. These trucks are powered with 150-hp. Cummins Diesel motors, equipped with "Wood" hoists, and are chain driven; they have 14 by 24-in. 20-ply tires.

The coal in this stripping is loaded with company-owned equipment, which consists of one No. 50-B and one No. 44-B Bucyrus-Erie electric shovels, each equipped with 24-ft. booms and 2-cu.-yd. dippers, and one Bucyrus-Erie electric dragline with a 55-ft. boom and a 2-yd. bucket. The coal is hauled by truck to a ramp on the southeast edge of the stripping, where it is transferred to standard-gauge railroad cars. This stripping produces an average of 35,000 tons of coal per month.

Fourteen Bucyrus-Armstrong No. 42-T 9-in. well drills are used to drill the rock-blasting holes, and the rock is taken off in benches up to 30 ft. in thickness. The holes are drilled in 15-ft. rows spaced 18 ft. apart and usually are deck-loaded with 40 per cent ammonia dynamite—except for wet holes, when gelatin dynamite is used. Fig. 4 shows nine of 14 No. 42 T-Bucyrus Armstrong 9-in. blast-hole drills.

Five thousand to 6000 lb. of dynamite is used in each round of shots. All drill holes are connected and detonated with Primacord. The sandstone in this stripping is extremely hard and while it cuts a little more easily in the drilling than the conglomerate, it is much harder to shatter.

The following data have been compiled from records kept on the shots fired and measured from Oct. 6, 1940 to July 30, 1942:

Average feet drilled, per drill, on a shift of 7 hr.	27
Cubic yards, per foot of hole drilled.........	7.6
Cubic yards drilled per drill shift............	205

In addition to the well drills, a tractor-mounted electric wagon drill is used in some locations on top of the roll, where it would be more difficult to set up standard well drills.

Auxiliary equipment consists of three D-8 caterpillar bulldozers, one 1000-gal. road sprinkler, one caterpillar Diesel No. 12 six-wheel road machine, one 800-gal. water tank truck to furnish water for the drills, and one oil truck.

A Bucyrus-Armstrong drill sharpener is part of the equipment in the well-equipped repair shop that the contractor maintains on the property. Major repairs to the equipment are made here, and steel bodies for the dump trucks are turned out in these shops. A Johnston furnace, to which is connected a Bristol pyrometer, is used for heat-treating the drills after they have been sharpened.

Ellangowan Stripping

This stripping is in the Ellangowan Basin 5 miles northwest of Mahanoy City and covers an area 500 ft. wide and 2200 ft. long on the Mammoth bed. The work is being done by Carey, Baxter and Kennedy, Contractors.

Before work was started, cross sections at 100-ft. intervals were constructed across the area and sufficient drilling was done to test the overburden. A general view of this stripping looking west is shown on Fig. 5. The pitch of the bottom slate might be noted on the right side of the cut, while the spoil bank which was side-cast with the dragline is seen on the left side.

On the right or north side of the stripping, the bed is solid and pitches south on an angle of 40° to about the center of the area, from whence it continues on a south dip of 20° and is over a pillar area that has caved. There is very little refuse in the coal bed structure, which is from 20 to 30 ft. thick.

There are approximately 2,000,000 cu. yd. of unclassified overburden in this stripping area, and 1,600,000 cu. yd. of refuse bank material—that is, breaker slate and ashes—which covers the north side or heavy pitch section. The overburden consists of 15 ft. of clay and an average of 50 ft. of very hard sandstone, which is interspersed with layers of shale. In some places conglomerate is encountered.

Three 33-cu. yd. La Tourneau Carryall Wagons, hauled by D-8 caterpillar tractors, are used to remove the bank material and some of the clay surface. Fig. 6 shows one of these Carryalls, with a maximum load. The D-8 caterpillar tractor in the rear is used as a pusher and is used only during the loading operation, and as soon as the loaded Carryall is moved out of the soft material the tractor used as a pusher returns to position ready to assist with the loading of the next Carryall. Two D-8 tractors are used for this purpose. The material is hauled a distance of 1200 ft. to a dump area. The three Carryalls are operated three 7-hr. shifts, or 21 hr. per day, and remove an average of 150,000 cu. yd. per month.

A No. 10-W Bucyrus-Monighan Diesel dragline, equipped with a 175-ft. boom and a 9-yd. bucket, is used on rock; this is changed to a 12-yd. bucket when the dragline is working on shale or softer material—most of which can be side-cast. This dragline handles from 90,000 to 210,000 cu. yd. of material per month, depending on the class of material removed.

The coal is loaded with a No. 43-B Bucyrus-Erie Diesel shovel, with a 24-ft.

boom and a 2-yd. dipper. Ten-ton Mack trucks, used to haul the coal, ascend out of the stripping cut on a maximum grade of 11 per cent and deliver the coal to a precleaning plant 1½ miles distant. An average of 20,000 tons of coal per month is produced from this stripping.

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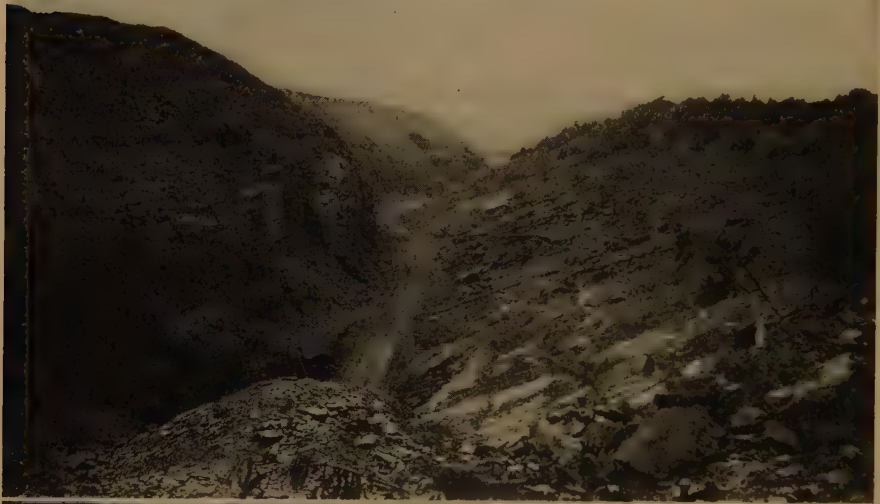

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Figs. 5 and 6.—Ellangowan stripping.

Rock holes are drilled in 15 to 18-ft. rows 15 ft. apart, depending on the character and depth of the overburden. These holes, which are staggered, are drilled with No. 42-T Bucyrus-Armstrong 9-in. well drills mounted on tractors.

Blasting is done with 0.6 to 0.7 lb. of 40 to 60 per cent ammonia dynamite or gelatin per cubic yard of rock. As a rule the holes are deck-loaded and Primacord is used for detonation.

Reevesdale Strippings

The Reevesdale strippings are on the south slope of Locust Mountain and on the north slope of Sharp Mountain. They extend along these mountains on both sides of Schuylkill Valley for a distance of 3 miles.

The work is being done by the Correale Construction Co. on an unclassified basis.

Three separate strippings are in operation at the present time, one being on the Mammoth bed on the Locust Mountain slope, another on the Orchard bed also on the Locust Mountain slope while the third is on the Primrose bed on the Sharp Mountain slope.

The Mammoth bed at this location is on the south dip and pitches 35° to 37°, and is from 12 to 15 ft. thick. With the exception of the chain pillar, this bed has been first mined and caved.

On the first cut, 18 ft. of clay and gravel overburden are removed, and on the second cut more wash and from 12 to 20 ft. of hard sandstone is blasted and removed.

Blast holes are spaced in 12-ft. rows 12 ft. apart; these are drilled with Bucyrus-Armstrong 6-in. gasoline-driven drills and the holes are charged with 40 per cent ammonia dynamite. All holes are connected with continuous Primacord, which is used for detonation.

The overburden is side-cast with a 6-W Bucyrus Monighan electric dragline that has a 175-ft. boom and an 8-yd. bucket. An average of 90,000 cu. yd. of material per month is moved by this machine, which is worked three 7-hr. shifts, or 21 hr. per day.

The coal is loaded into 11-cu.-yd. Sterling end-dump trucks with a 2-cu.-yd. Osgood Diesel-powered shovel and hauled to a cleaning plant 1½ miles distant.

Stripping on the Orchard bed was started at a central point on the outcrop with one dragline and as soon as the pit was opened sufficiently a second dragline was added; all overburden was side-cast. The south wall of the stripping is on 90° dip while the north wall is on a south dip of 50°. Fig. 7 shows this pit looking west from the eastern end. After removing 30 to 40 ft. of clay, the coal bed was exposed and measured 65 ft. across. There is very little refuse in the bed and the coal is a very good quality. One hundred feet below the surface the bed narrowed down to 20 ft. thick.

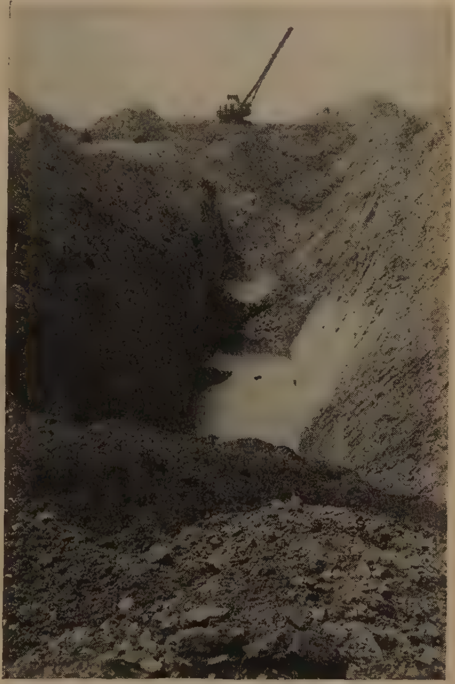


FIG. 7.—REEVESDALE STRIPPING.

One No. 4161 Marion electric dragline with a 100-ft. boom and a 4½-yd. bucket, and one No. 4101 Marion electric dragline with a 75-ft. boom, and a 3-yd. bucket are used on this job. These draglines are also used to load the coal into 11-yd. Sterling trucks, and it is hauled one mile to the cleaning plant. As this stripping was in operation only a comparatively short time no statistics on the performance of the equipment was available.

The third pit that is part of the Reevesdale operation is on the north slope of

Sharp Mountain on the outcrop of the Primrose bed. The bed pitches from 60° to 90° north, and is from 30 to 80 ft. thick; in some places it is divided by rock intrusions and faults. The outcrop is covered with 4 to 15 ft. of clay and gravel; on the first cut this is cast on the upper or south side of the pit; on the second cut, in addition to the wash, 36 ft. of sandstone and slate are removed; this is cast to the north side of the pit with a 5-W Bucyrus-Monighan Diesel dragline. This dragline has a 110-ft. boom and a 6-cu.-yd. bucket, and moves 60,000 to 70,000 cu. yd. of material per month. The rock is drilled with Bucyrus-Armstrong 9-in. Diesel well drills, the holes being spaced in 18-ft. rows 18 ft. apart and charged with 40 to 60 per cent ammonia dynamite. The Marion 39-A Diesel dragline is used to load the coal. This machine has a 75-ft. boom and a 1¾-yd. bucket; it recovers the stripped coal to a depth of 80 ft. below its base.

In this stripping work, as the loading of one truck is about completed another backs in beside it, so that there is little or no time lost with the dragline. There is just enough room in the Primrose pit to allow the bucket to pass the side walls as it swings around in the loading cycle.

The Reevesdale strippings produce an average of 50,000 tons of coal per month.

Fig. 8.—Central cleaning plant, Reevesdale stripping.

This is all transported in trucks to a central cleaning plant (Fig. 8) where all the large-size refuse is removed and the plus 6-in. coal is broken down so that the maximum size will not be greater than 6 in. After passing through the cleaning plant, the coal passes to a 42-in. belt conveyor, which carries it across the state highway to the railroad cars, a distance of 500 ft. It is then transported to the St. Nicholas breaker for preparation. Adequate equipment is provided for the proper maintenance of good roads around the strippings, and a well-equipped repair shop is maintained on the property.

Lehigh Navigation Coal Company

Summit Hill Stripping

The largest stripping operation in the anthracite field is that of the Lehigh Navigation Coal Co., Inc., at Summit Hill, Carbon County. This stripping has been in continuous operation since 1928. At the present time, it takes in an area 1200 ft.

wide by 4000 ft. long and has a maximum depth of 400 ft.; it is on the Mammoth bed in the Summit Hill Basin. Fig. 9 shows a cross section through the Summit Hill Mammoth bed stripping 400 ft. east of the present working face. The dotted line through the section shows the elevation of the top of the coal as originally in place. The basin pitches about 15° to the east; the greatest depth at which coal is now recovered is 400 ft. below the original surface.

To date 20,000,000 cu. yd. of overburden and coal-bed material have been removed, including 5,000,000 tons of coal that have been recovered from this operation to date.

The normal thickness of the south dip of the Mammoth bed is 55 ft. but on the north dip, owing to folds and other irregularities, it is 120 ft. thick in some places. A large part of the area of this stripping has been first-mined and, because of the development necessary for underground mining, connecting cross tunnels were driven, and these tunnels together with the old first mining provide good drainage for the stripping.

A mine fire that started 83 years ago in the Summit Hill Basin is still smouldering in some small isolated parts of the stripping.

Seven 6-in. Loomis-Clipper and Cyclone churn drills, driven by electricity and gasoline, are now used to drill the blasting holes although there have been times when 15 drills were required. The drill holes average 40 ft. in depth and are drilled in 12½-ft. rows 12½ ft. apart, staggered. As many as 40 or 50 holes are fired at one time and deck-loaded with 40 or 60 per cent gelatin. The quantity of explosive used averages 0.65 lb. per cu. yd. of rock blasted. In hot areas, where the temperature of the rock ranges from 110° to 180°F., water often is used to cool the holes before the blasting charges are placed. In holes of abnormally high temperatures, 40 per cent gelatin wrapped with an asbestos covering is used.

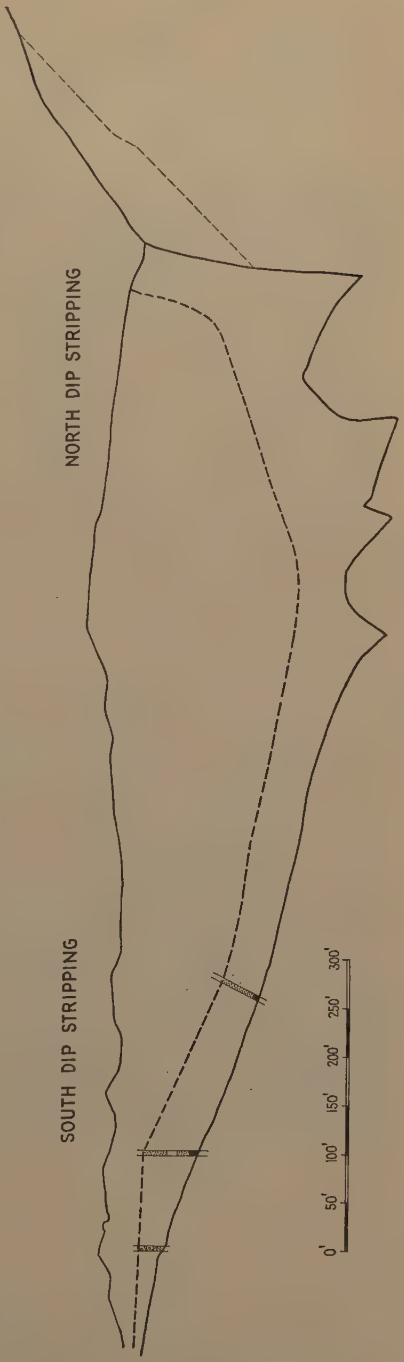


Fig. 9.—Summit Hill Stripping.

North-dip Section.—In the north-dip section the bed pitches from 20° to 90°. The greater part of the overburden has been removed. In Fig. 10 are shown the different grades on which the steam locomotives are operated in order to haul the coal out of this section.

One 120-B Bucyrus-Erie shovel with a 30-ft. boom and a 4-yd. dipper, and one No. 4161 Marion dragline with a 110-ft. boom and a 4-yd. bucket, both machines electrically driven, are used to load the rock and coal. The equipment used depends on the location of the material to be loaded. Coal or rock above the elevation of the railroad track is loaded with the shovel, while the material below the level of the railroad is loaded with the dragline. At the time of this survey the dragline was 30 ft. below the level of the railroad track and was excavating to a depth of 70 ft. below its base, making it possible to recover coal 100 ft. below the lowest track elevation (Fig. 11). Usually the coal is loaded into standard-gauge railroad hopper cars, except coal from burned areas, which is partly mixed with rock; this is loaded into 30-cu.-yd. side dumpers and sent to the cleaning plant. Rock is transported to the rock dump in 30-cu.-yd. side-dump cars made up in two-car trips, hauled by 50 and 60-ton Heisler geared steam locomotives. The branch at the top of the stripping pit is reached by back-switching to attain proper grade and the haul is approximately 2 miles.

A common track grade of 3 per cent is maintained as nearly as possible, although it is necessary to use up to 5 per cent grade for short distances. The coal cars are assembled at the top of the stripping in trains of 20 cars and transported to the breaker, 2 miles distant, with standard side-rod locomotives.

The rock cars are switched out to the rock dump, which carries an ascending grade of 2 per cent. A rock-spreader car is used to shift the rock away from the track and when a sufficient berm has been established the track is moved to the edge of the bank with a Nordberg track shifter.

South-dip Section.—In the south-dip section of the Summit Hill stripping one 120-B Bucyrus-Erie electric power shovel, with a 30-ft. boom and a 4-yd. dipper, is used to load the overburden, and one No. 43-B Bucyrus-Erie electric shovel, with a 24-ft. boom and a 2-yd. dipper, is used to load the coal. Coal and overburden are both hauled in 8 and 10-cu.-yd. end-dump Mack trucks powered by Cummins Diesel motors. The coal is hauled to a ramp at the rim of the stripping, where it is transferred to standard railroad cars. The overburden is hauled out of the stripping to a spoil dump, although back filling is done whenever possible.

An average of 106,000 cu. yd. of material per month, including 52,000 tons of coal, is excavated from the Summit Hill strippings.

Machine shops to repair the equipment, including the steam locomotives, are maintained at the stripping.

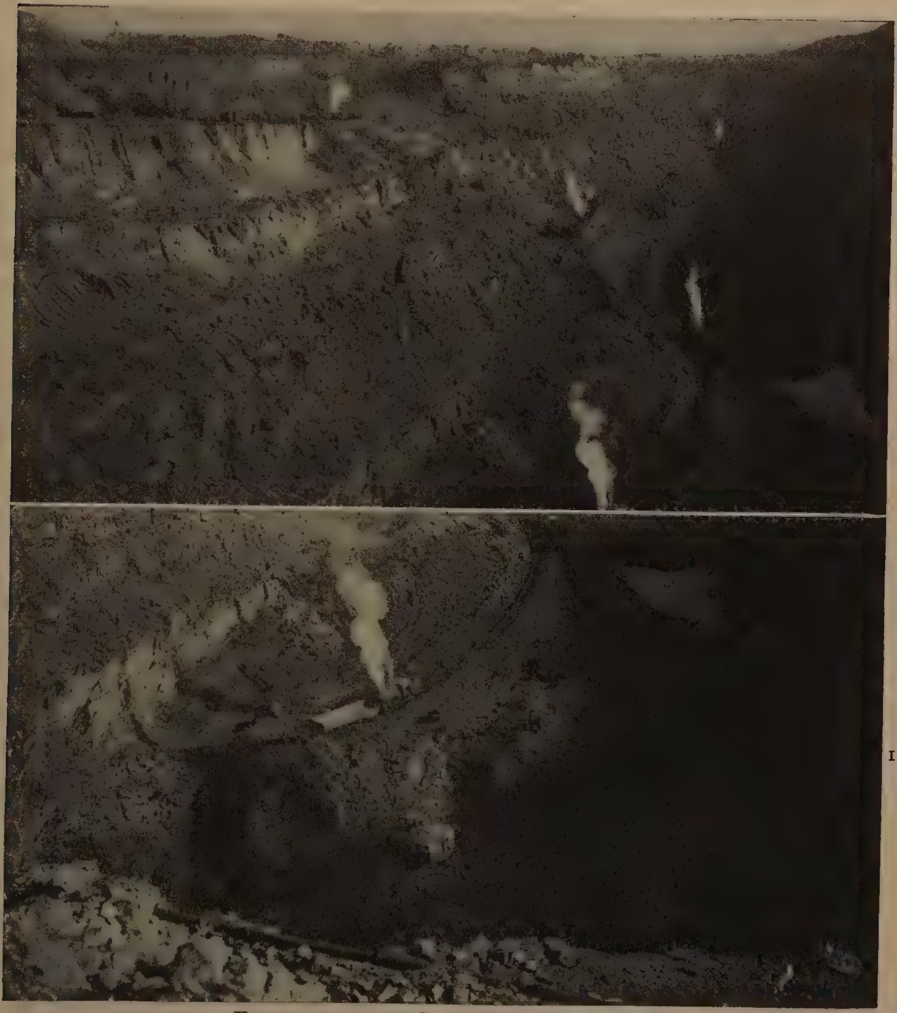
Coaldale Stripping

The work on the Coaldale stripping is being done by the Central Pennsylvania Quarry and Stripping Co. This stripping, on the south dip of the Mammoth bed in the Panther Creek Valley, is one of a series of stripping operations on this bed that have made an almost continuous cut along the mountain for a distance of approximately 3½ miles.

The average pitch of the bed is 50° south and the coal is 40 ft. thick. A typical cross section of the coal measures in the Panther Creek valley is shown in Fig. 12. The proximity of the four lowest workable beds is shown in this cross section.

The wash is 12 ft. thick and, as none of the material could be cast on the high or north side of the pit because of the slope, it was necessary to haul all of the crop overburden in the first cut in motor trucks before the chain pillar could be removed.

About 38 ft. of shale and sandstone, in addition to the wash, is being removed in the second cut. Most of this can be sidecast, as the next bed (Holmes) farther down the mountainside had been stripped before the stripping on the Mammoth bed was started.



Figs. 10 and 11.—Summit Hill stripping.

Usually, fifty 6-in. rock-blast holes, which on the second cut average 50 ft. in depth, are charged at one time. The holes are drilled in 15-ft. rows 15 ft. apart and are deck-loaded. The total charge in these 50 holes contains about 10,000 lb. of 40 per cent ammonia dynamite, or 0.60 lb. of explosives per cubic yard of rock blasted.

A Marion No. 125 electric dragline with a 100-ft. boom and a 3-yd. bucket is used to remove the overburden. One Marion No. 371 electric shovel having a 24-ft. boom and a 1¾-cu.-yd. dipper is used to load the coal into trucks, which haul it one mile to the Coaldale breaker, to be prepared for market.

The dragline moves 600 cu. yd. of overburden per 7-hr. shift, while the No. 371 shovel loads 800 tons of coal during the same period.

50 per cent gelatin. In one charge the holes averaged 57 ft. in depth and 31,000 lb. of explosive were used in 42 holes; or 0.7 lb. of explosive per cubic yard of rock blasted. The holes were filled to within 15 ft. of the top and were stemmed with clay. This blast was detonated with Primacord.

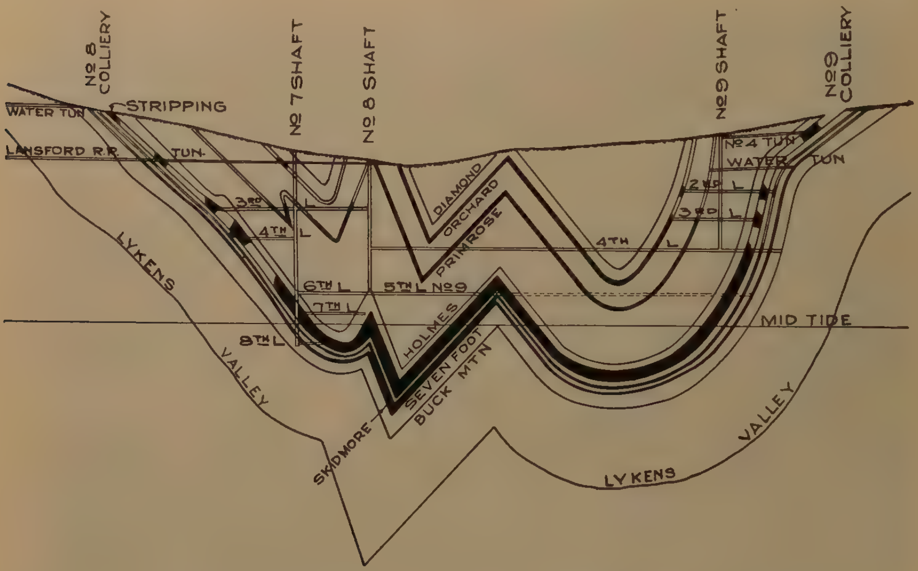


FIG. 12.—SECTION OF COAL MEASURES, PANTHER CREEK VALLEY.

ST. CLAIR COAL COMPANY

This company is operating two separate strippings on the south dip of the Mammoth bed about 1½ miles east of the town of St. Clair. The work is being done by J. Robert Bazely, Inc., Contractors, on a classified yardage contract.

Pine Forest Stripping No. 50

In the first stripping, known as Pine Forest No. 50, the average pitch of the bed is 20°, and the bed is from 24 to 30 ft. thick. The overburden consists of 6 to 8 ft. of clay and gravel and from 50 to 70 ft. of shale and sandstone.

Blast drill holes are drilled with 6-in. Loomis electric well drills mounted on tractors. The holes are spaced in 16-ft. rows 16 ft. apart and are charged with

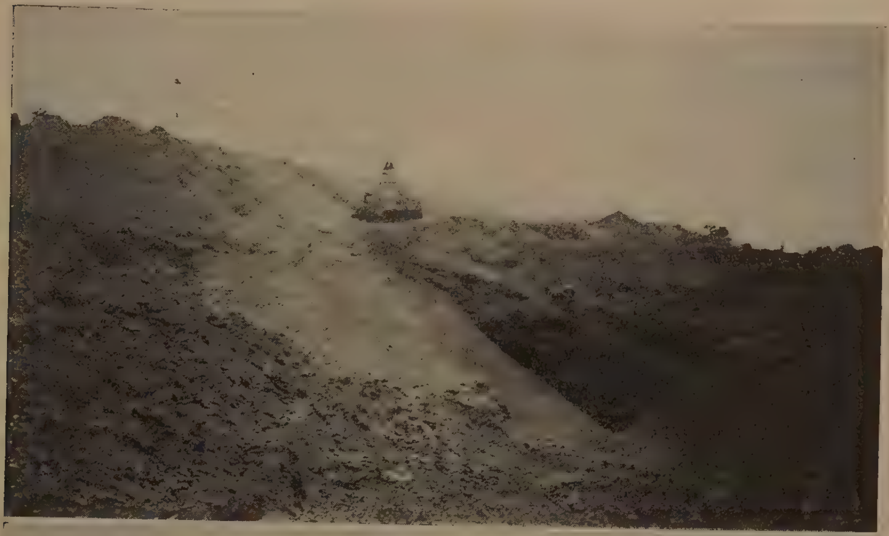
The rock is loaded by a No. 52-B Bucyrus-Erie Diesel shovel with a 24-ft. boom and 2½-yd. dipper. Nine-yard Euclids and 11-yd. Mack trucks haul the rock to a dump 1000 ft. from the pit. Some back filling was done where that was possible.

Owing to the confined area of the pit it was possible to ramp only part way down into the pit, to a point just above the coal. This made it necessary to load the coal with a dragline—a No. 52-B Bucyrus-Erie with a 65-ft. boom and a 2½-cu.-yd. bucket. The 9-cu.-yd. Euclid end-dump trucks that are used to haul the coal ½ mile to a coal pocket are equipped with Diesel motors. The coal is transferred at the pocket to narrow-gauge cars and hauled by steam locomotives 4000 ft. to the breaker.

The stripping is worked on two 7-hr. shifts per day. The shovel averages 130 cu. yd. per hour on rock and the dragline 60 cu. yd. per hour on coal.

Pine Forest Stripping No. 55

The Pine Forest No. 55 stripping is on the south dip of the Mammoth bed and covers an area 300 ft. wide and 2200 ft. long. The minimum pitch in this stripping is 30°; the maximum, 50°, on the eastern end. The coal along the outcrop was stripped a number of years ago and the present stripping is a continuation, or extension, of the old stripping, over an area that was first-mined and affected by a squeeze. The original height of the bed was from 24 to 26 ft. but owing to the squeeze it was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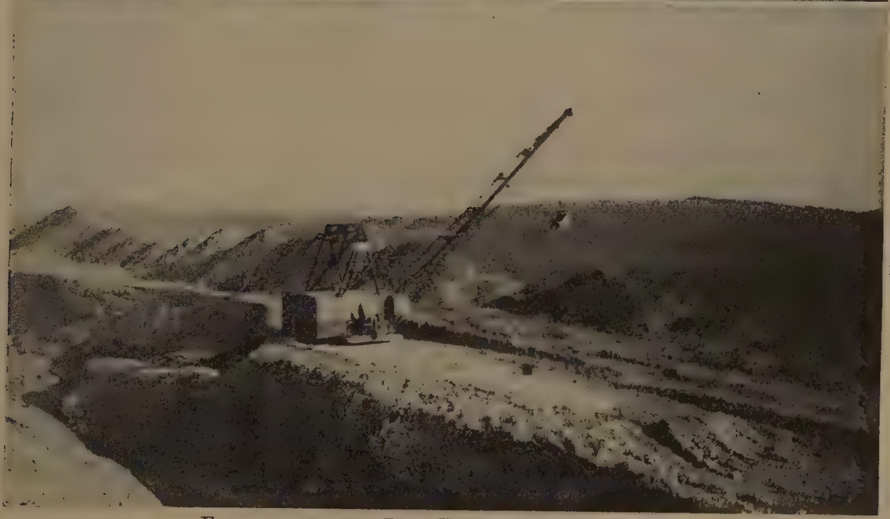
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FIGS. 13 AND 14.—PINE FOREST STRIPPING NO. 55.
Fig. 13, north side; Fig. 14, south side.

crushed down to 12 ft. This bed contains few or no impurities. The overburden is from 20 to 70 ft. thick, of which 18 ft. is clay and gravel and the remainder is shale and sandstone. One of the unusual features of this stripping is that when the squeeze affected this area the sandstone was broken up sufficiently to make blasting of the rock unnecessary.

A 9-W Bucyrus Monighan electric dragline is used to remove the overburden. It has a 200-ft. boom and an 8-yd. bucket; it is one of the largest units of stripping equipment in the southern field. The motor-generator set on this machine consists of a 2200-volt a.c., 400 kva., 450-hp. motor, which drives three generators on the same line shaft. The generator for the hoist motor is a 900 amp., 225 kw., 250-volt machine, and the generator for the swing motor is 270 amp., 135 kw., 250 volts. Another small 125-volt generator provides power to drive the air compressor and light the machine. The dragline is operated three shifts of 7 hr. each, or 21 hr. per day and moves an average of 256 cu. yd. per hour.

Because of the long length of the boom, it was possible not only to cast much of the material on the first cut on the north side over into an old stripping pit, but also to pile the spoil on the upper slope of the mountain after the old stripping pit was filled (Fig. 13).

After the cut was completed on the north side the dragline was moved over to the south side and the wash was removed, the cut being 125 ft. wide. The material was cast as far as possible (see right side of Fig. 14) and in this manner space was left for the rock in the same cut that was excavated later.

The 7-yd. trucks used to haul the coal to the pocket, 4000 ft. distant, are loaded with a No. 77 Lorain six-cylinder Diesel shovel with a 22-ft. boom and 1½-cu.-yd. dipper.

The trucks ramp down into the bottom of the pit; by using this small shovel it is possible to avoid loading most of the broken rock that caved into the breast openings and sometimes is mixed with the coal.

The contractor maintains a well-equipped repair shop on the property for making minor repairs. Drills are sharpened and the major repairs are made at the contractors' general repair shop at Port Carbon, 4 miles from the stripping.

Auxiliary equipment consists of two bulldozers, one road patrol, one gasoline truck, one water truck, two air compressors for small rock drills, and six well drills.

ACKNOWLEDGMENTS

The authors wish to thank the officials of the following corporations and contractors for information furnished in connection with this paper:

The Philadelphia and Reading Coal and Iron Co.; the St. Clair Coal Co.; Lehigh Navigation Coal Co.; East Bear Ridge Collieries Co.; Contractors Carey, Baxter and Kennedy; Dick Construction Co.; Capperel Construction Co.; Correale Construction Co.; Central Pennsylvania Quarry, Stripping and Construction Co.; J. R. Bazely Inc., and Schumacker and Schultz.

Their appreciation is extended also to the following for their fine cooperation: Mr. Roos and Mr. McCullough, of the Philadelphia and Reading Coal and Iron Co.; Messrs. J. F. K. Brown and C. W. Wagner, of the Hudson Coal Co.; Mr. C. B. Elsasser, of the Edison Anthracite Coal Co.; and especially Mr. A. E. Coddington, of Carey, Baxter and Kennedy, and Mr. Ralph Sterner, of the Lehigh Navigation Coal Company.

DISCUSSION

Charles Enzian,* Hazelton, Pa.—The authors of this paper are to be complimented on the manner in which they bring up to date an important yet simple, efficient, and economical method of mining anthracite. J. B. Warriner and H. H. Otto covered the subject up to about 1931, but the vast changes in operating methods, which required larger capacity and more efficient equipment, and the improved methods in transportation of the waste material and coal, have made the present time a most opportune marking off of the stages of progress.

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* Consulting Mining Engineer.

Very few engineers now living have had the opportunity to study the methods of stripping anthracite overburden from the pick-and-shovel and wheelbarrow or dump-cart stage to the present large-capacity shovel and dragline excavators, and the truck, or euclid, or turneau self-loading and tractor hauling equipment now almost exclusively used in the Wyoming, or northern field, and to a large extent in the eastern and western middle field, and to a lesser extent in the southern field, of the anthracite region.

It is regrettable that the authors did not find it practicable to discuss in appropriate detail the cost information of the several projects; after all, these data are the real measure of success in this method of mining. It is apparent to all engineers that the cost of stripping the hard, crystalline, massive dark gray sandstone overburden of the Buck Mountain, or red-ash bed is considerably higher than the cost of removing the sandy shale and medium hard, laminated sandstone overburden of the Top Split Mammoth or overlying beds.

During the past 20 years the writer has had occasion to study intensively the methods and analyze the costs of stripping operations in the strata overlying all of the anthracite beds. The costs adjusted to the present wage scale would indicate that large-scale efficient operations in reasonable thickness of coal—say 5 to 10 ft. and an overburden ratio of about 6 to 1—would vary from about \$2.50 per ton of coal recovered from solid formation, and a cost of about \$3.25 per ton of coal recovered in first-mined areas.

The stripping cost of the lower or harder sandstone formation overburden would be appreciably higher, depending upon such important factors as casting space, thickness of coal and formation conditions, and the character and hardness of the overburden removed.

Use of Tubing and Blowers for Auxiliary Face Ventilation

By Raymond Mancha,[*] Member A.I.M.E.

(New York Meeting, Febuary 1944)

The use of blower fans and tubing for auxiliary face ventilation is a somewhat controversial subject. Most authorities agree upon the advantages of this system over the maintenance of line brattice for dead-end ventilation. They also recognize the system's disadvantages. However, there is considerable disagreement as to the proper relation of the advantages to the disadvantages.

It is the purpose of this report to present the opinions of four authorities of diversified occupation and permit the reader to form his own opinions on the subject.

Observers

Carel Robinson is a mining engineer consultant of wide repute, who has a background of operating experience.

Thomas G. Fear is a mine operator whose present position is that of chief engineer for the Hanna Coal Co. at St. Clairsville, Ohio.

E. H. Denny is chief of the Coal Mine Inspection Division of the U. S. Bureau of Mines, and is a very able engineer whose past experience well qualifies him to appraise this problem.

Cloyd M. Smith, editor of the periodical *Mechanization*, is a ventilation consultant. For a number of years he was a member of the Mining Department of the University of Illinois and has studied mine ventilation for many years.

Opinions

It is Mr. Robinson's opinion that closely spaced breakthroughs, although conducive to low gas accumulation past the last open crosscut during the driving of entries, are difficult to maintain and are constant sources of fugitive air. Jute or canvas brattice is suitable for entry driving with

Table 1.—*Figures from Federal Reports*

Federal Inspection District	State	Number of Inspection Reports Studied	Mines Using Auxiliary Fans		Number of Auxiliary Fans Used
			Number	Per Cent	
A	Western Pennsylvania, Ohio, Northern W. Virginia, Maryland	381	47	12.3	252
B	Eastern Pennsylvania Anthracite	27	18	66.7	258
C	Southern W. Virginia, Virginia, Eastern Kentucky	197	35	17.8	156
D	Tennessee, North Carolina, Alabama	47	14	29.8	59
E	Indiana, Western Kentucky, Illinois	178	30	16.9	230
F	Michigan, North Dakota	9	3	33.3	7
G	Arkansas, Oklahoma	11	4	36.4	18
H	Colorado, Utah, Washington, Wyoming	50	26	52.0	204
Total..		900	177	19.7	1,184

Report of Ventilation Committee, Coal Division A.I.M.E. Manuscript received at the office of the Institute May 26, 1944.

[*] Chairman Ventilation Committee, Coal Division A.I.M.E.; Manager, Mine Ventilation Department, Jeffrey Manufacturing Co., Columbus, Ohio.

closely spaced breakthroughs, but blowers and tubing are required with breakthroughs on long centers.

Mr. Robinson points out that dust hazard is aggravated by mechanized mining, but usually can be controlled in adequately ventilated gassy mines with line brattice. In nongassy mines, however, blowers and tubing prove most effective for rapid removal of dust and smoke.

Mr. Robinson maintains that prejudice against blowers and tubing caused by improper use should not justify blanket condemnation of those properly installed and maintained.

On the other hand, Mr. Fear contends that blowers and tubing should be outlawed because of recirculation, interrupted operation, a feeling of false security and their past record of operation.

Mr. Denny refers to 900 Federal Inspection reports and 366 Bureau of Mines disaster reports, citing the low percentage of blowers with permissible electrical equipment, also the prevalence of intermittent blower operation, and the high percentage of installations that constitute definite fire hazard (Table 1).

Five of the 1184 auxiliary fans in use were operated by compressed air and the remainder by electricity.

Only 40 of the 1184 auxiliary fans had a type of motor and control rated by the Bureau of Mines as permissible. This permissible rating means that such motor and control if properly maintained would not ignite gas if inadvertently operated in an explosive mixture of methane and air. Over 95 per cent of all the auxiliary fans reported in use had nonpermissible motors and controls, and constitute potentially dangerous sources of ignition if operated in explosive mixtures.

Of the 1184 fans, 232 were reported as operated continuously, and 936 intermittently. The facts as to continuous or intermittent operation could not be determined for the remaining 16 fans.

"Intermittent operation" usually means operation only during the time the regular shift is working and cessation of fan operation during the off shift. The term also includes operation only when work is going on at the particular face ventilated by means of the fan and tubing.

Also, all auxiliary fans, whether listed as being operated continuously or intermittently, are subject to stoppage because of local or general mine power interruptions. In some cases the normal ventilation of the working faces by crosscuts and brattices is considered adequate by the mine managements and the fans and tubing are used intermittently for the more rapid dispersal of the smoke and gases developed by blasting.

TABLE 2.—*Gas Ignitions Causing Fatal or Serious Injuries during the Fiscal Year* 1943

State	Number of Explosions	Mines Using Auxiliary Fans	Men Killed	Men Injured
Alabama	3	0	12	8
Arkansas	4	2	5	9
Colorado	1	1	3	0
Illinois	2	0	1	7
Kentucky	6	2	10	15
Montana	1[a]	0	74	0
Ohio	3	1	7	1
Oklahoma	2	2	7	10
Pennsylvania	3	1	1	15
Tennessee	1	1	10	2
Virginia	1	0	0	4
West Virginia	3	0	21	2
Total	30	10	151	73

[a] Booster fans used.

Mr. Denny emphasizes the number of cases of air recirculation caused by improper location of blower. He protests against improper tubing installation, permitting constrictions and leakage, as well as instances of blower use without tubing or with tubing terminating too far from the face. Such factors account for the amount of poor face ventilation referred to in the inspection reports.

Mr. Denny states that whereas proper company rules regulating the installation and maintenance of blowers and tubing could prevent recirculation, evidence indicates that such regulation is lacking or is unobserved.

Mr. Denny points out that in 52 instances of gas ignition and mine explosion resulting in injury or loss of life, misuse of blowers and tubing contributed to gas accumulation and ignition, and the use of the equipment was accompanied by inadequate face ventilation, bad air courses, inadequate air at the main fan, or other ventilation short cuts to speed production or cut costs. Mr. Denny cites ten examples of disaster in which blowers and tubing are claimed to have played a contributing part (Table 2).

Since the percentage of inspected mines using blowers (19.7) is less than the percentage of blown-up mines using blowers (33⅓) Mr. Denny concludes that the use of blowers and tubing may present an increased hazard.

Mr. Denny next quotes the Bureau's 1927 recommendations regarding the use of blowers, which relegate their use to emergency work such as driving a cross heading in rock to make connection with distant workings and where distances are excessive for line brattice, and he proceeds to stipulate proper methods of installation and operation. Mr. Denny also refers to the paper by Greenwald and Howarth,* which outlines proper installation precautions for blowers and tubing.

Mr. Denny concludes that the fire hazards due principally to the manner of installation or use of blowers and tubing, plus the examples of explosions and ignition cited where use of these fans is said to have played a contributing part, justifies disapproval by the Bureau of the use of such equipment.

Mr. Smith first differentiates between primary and auxiliary types of ventilation, explaining the greater dependence upon auxiliary ventilation in metal mines than in coal mines, which he explains to be the result of better ventilation in the latter than in the former.

He recognizes the necessity of both safety and effectiveness in the use of auxiliary blowers and tubing in coal mines and states that these objectives are compatible. He emphasizes the necessity of blower location so as to avoid air recirculation and the handling of dust or gas-contaminated air. Also, he insists upon the use of approved and permissible electrical equipment and continuous blower operation and emphasizes the importance of proper precautions, such as suspension of tubing, freedom from kinks, bends, etc.

Mr. Smith explains the advantages of blowers and tubing for driving entries and maintaining crosscuts on long centers, also for reducing the number of crosscuts between conveyor-mined rooms. He again cautions against placing room blowers so as to permit recirculation or dust contamination.

OBSERVATIONS

Whereas these four reports express differences of opinion as to the relative advantages and disadvantages accompanying the use of tubing and blowers for auxiliary face ventilation, they are in complete agreement as to certain basic precautions that should be employed when such equipment is used. For example, all observers agree that such blowers should be powered with permissible Government-approved motors and that proper ground connections should be employed. Furthermore, the blowers should be kept in continuous operation, to avoid high concentration of gas deposits during shutdown periods. It is important also that the blower be placed on fresh air in such a manner as to exclude all possibility of recirculation of air already handled by the blower.

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* H. P. Greenwald and H. C. Howarth: Recirculation of Air and Mine Gas Caused by Auxiliary Fans as Used in Coal Mines *Trans.* A.I.M.E. (1928) **76**, 164.

The main justification for the use of tubing and blowers for ventilating dead ends is where the installation and maintenance of line brattice is rendered impractical, either because of long distance desired between crosscuts or the operation of mechanical equipment with space limitations.

By presenting the diversified opinions of four competent observers it is felt that the material contained in this report will prove of most value to the mine operator who is interested in the use of this particular class of equipment.

DISCUSSION

(*C. T. Hayden presiding*)

L. E. YOUNG,* Pittsburgh, Pa.—Mr. Mancha has presented a paper that should stimulate much discussion and the opinions on this subject that have been expressed by four members of the Institute cover the matter quite thoroughly. All of us are acquainted with the recommendations, standards, and decisions of the United States Bureau of Mines and with the mining laws of the several states in which we operate. In some instances, the use of tubing and blowers is prohibited entirely, while in others it may be permitted by special action of a district mine inspector.

Mining conditions and mining practice change, and with these changes laws and regulations may reasonably be revised, if personal hazards are not increased. Owing to the depletion of the better seams of coal, we will be obliged to mine under different and more difficult conditions than those for which most of our existing mining laws and regulations were formulated. In some instances, the mining hazards will be increased, but an intelligent approach to the problems may make economic mining possible.

For example, we are now mining thinner coal seams, which cannot be mined economically by hand. If a number of established mining communities are to be perpetuated, it will be necessary to install low-height equipment, which may be difficult to move; to do less brushing of panel entries and rooms; and to cut and load several times a shift in each working place. In such concentrated mining, the removal of powder smoke quickly is essential. When gas is cut in clay veins, under existing mining laws and regulations, the gas must be removed before even permissible electrical mining machinery may be operated. Unless this gas is removed promptly, it creates a serious hazard, and at the same time prevents face work on schedule. The scheduling of operations is essential for the success of concentrated mining.

In a well-ventilated mine, the volume of air directed into a working panel is normally more than adequate to comply with the legal requirements; but when a clay vein is cut and an unusual flow of gas is encountered, an increased volume of air is required to sweep out and dilute this gas. Until this gas is removed, all electrical machinery must be stopped.

It has been proposed that tubing and blowers shall be used only to supplement the already existing ventilation, which is presumed to be adequate for normal conditions. In my opinion, tubing and blowers should not be permitted as a substitute for well-ordered ventilation, but from an engineering standpoint it ought to be possible to design and build portable permissible blowers that will supplement existing ventilating facilities, and remove gas and smoke quickly from a particular face and working section. Such blowers should not be operated except for the few minutes necessary to remove powder smoke and when gas is produced or accumulates in excess of the safe working limits.

If we are going to expect men to work efficiently, we must remove powder smoke quickly. Some of the most modern and efficient installations keep the cutting machine in the same working place and the face may be cut five or more times per shift. Obviously, good ventilation must be provided to permit the face men to work. If gas is present, there may be considerable delay in getting it out.

This is a real challenge to the safety engineers of the industry and deserves the constructive attention of the Bureau of Mines and the State Departments of Mines, as well as the manufacturers and operators.

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* Consulting Engineer.

J. Pursglove, Jr.,* Pursglove W. Va.—Dr. Young seems to have approached this problem as an engineer, and I have tried to approach it as an operator of a gaseous coal mine in northern West Virginia. Mr. Mancha wrote to a number of people and passed on their ideas in his paper. I thought I would do the same thing; so I wrote the Chief of the Department of Mines in West Virginia, Jesse Redyard. The West Virginia Department of Mines has for several years been absolutely opposed to the use of blowers. It will not give a permit for the use of one, and probably will put a danger board over the pit mouth or shaft of any mine that uses one.

I sent the Chief of the Department of Mines of West Virginia a copy of this paper. He said it was so complicated that he did not have a chance to study it in detail, but I did get a letter from him, in which he says: "I have not had the time to study Mr. Mancha's paper, but he is a well-known and outstanding engineer and I am sure he has covered the subject of auxiliary face ventilation very thoroughly, and his paper will no doubt be of interest to those interested in mine ventilation. As you know, the Department of Mines of West Virginia is not in favor of the use of blower fans for ventilating purposes." I tried to get him to come to the meeting and substitute for me, but he would not do that, so he let me come and bring along his letter.

When I was asked to discuss this paper I said that I did not know whether I was qualified to do so, because I had absolutely no experience with blowers and did not know anything about them. In thinking about the matter, however, it occurs to me that in gaseous mines, with mechanical loading, where the time or element of liberation has been increased two, three, or four times what it was under hand loading, it is not possible to deliver enough air up into a working face that is liberating gas, with a blower; unless, of course, the blower is about as large as the outside fan.

The Chief of the Department of Mines of West Virginia called me on the telephone and added a little bit to his remarks. He said: "Of course, you know that the rules of the Department of Mines of West Virginia are that there must be a minimum of 6000 cu. ft. of air per minute to the last breakthrough—that is the minimum—and that there shall be enough to render any harmful gases harmless." Well, in our mines, in places, 15,000 or 20,000 cu. ft. per minute is required at the faces; and then some places must be shut down for a few days if any serious gas bleeders occur, to let the gas bleed itself out. It is not possible to work them even with 20,000 cu. ft. per min. at the face.

Now, I am offering that as a question: How would you get a blower with enough capacity to take care of a situation of that kind? I do not see how you would do it.

As I see it, a blower close to the working faces is just one more additional hazard—and there are plenty of hazards without blowers in these gaseous mines. It is all right to say that from an engineering or theoretical standpoint *if* it is all maintained in "apple pie" order, everything will be all right, but we all know that things cannot be kept in that kind of condition.

The wiring up to a motor is a hazard; and, assuming that you can have a permissible blower, and it is in "apple pie" order when installed, it would not be in that condition after the first week. Very few mines in the country have machinery that is actually permissible according to the U. S. Bureau of Mines standards. It is all right when it is bought, but before long it is not perfectly permissible; and, as I see it, one more hazard is being added close to the face, permissible or not.

As far as smoke removal is concerned, in a mine that makes much gas the brattice cloth must be kept up close to the face, anyway. So much air sweeps around the end of the brattice cloth (it is necessary because of the quantity of gas liberated), and with 10,000 or 15,000 ft. at the face a blower is not needed to sweep any smoke away; it disappears quickly.

Not knowing very much about the subject, but hearing other people's views, it occurs to me that a blower is just one more complication and hazard at the face.

R. D. Hall,* New York, N. Y.—Too much stress has been laid on the hazards of auxiliary fans, because of the newness of their arrival.

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* President, The Pursglove, Coal Mining Company.

* Engineering Editor, *Coal Age*.

The ancient device—the line brattice—which they replace is the more suitable for criticism. Such fans: (1) take their air from the heading, unlike the line brattice, which receives much of its air from passages along the goaf (Figs. 1 and 2);* (2) discharge the air near the roof, where it can remove methane, and not near the floor, where it will disturb coal dust; (3) mix the methane with the air by the turbulence of their effluents, thus making it travel to the return and not collect in pockets; (4) increase air volume by relieving load on main fan.

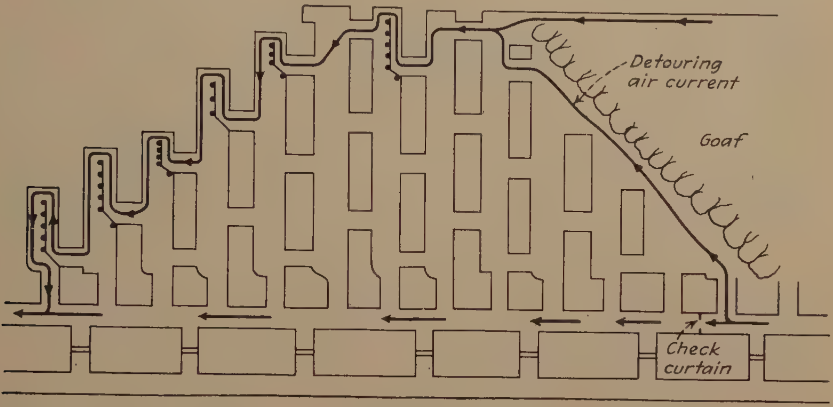

FIG. 1.

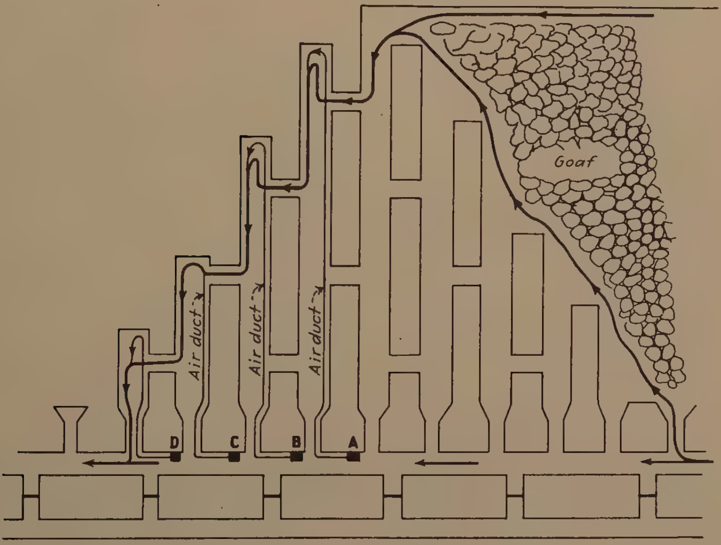

FIG. 2.

With line brattice: (1) air is drawn more from edges of goafs than from the heading and is filled with methane before it reaches the first advancing face; (2) methane is not so readily removed as with fans, because the air is not concentrated near the roof; (3) coal dust is disturbed; (4) air volume is decreased by increased resistance due to the narrowed passages to be traveled between line brattice and coal ribs; (5) material of brattice often is torn down and damaged, allowing air to short-circuit face and accordingly permitting methane to gather.

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* As long as the auxiliary fans are running, they tend to keep the air traveling within the heading and out of the rooms and caved areas adjacent thereto until, however, that air arrives at the mouth of an auxiliary fan and is whisked to the end of a working place. Thus, the caved areas and rooms outbye the fans are ventilated most actively when the latter are idle.

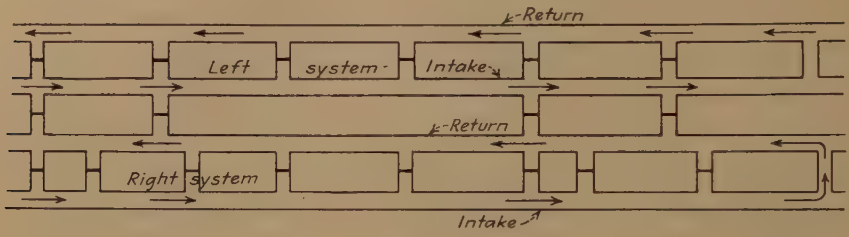

FIG. 3.

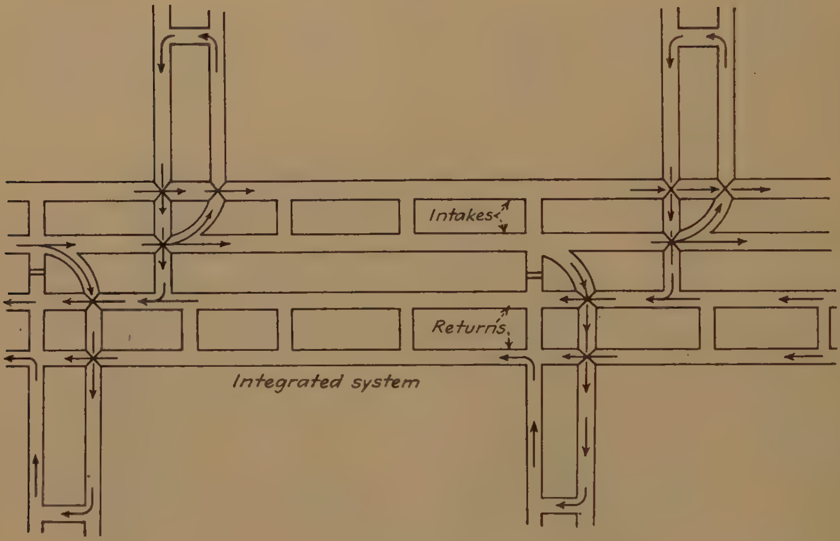

FIG. 4.

With two auxiliary fans in a heading, it is said that air from one leaves the first room and is re-used by the second and that only one receives fresh intake air, but the air travels from room to room and probably with long rooms never returns to the heading by the room through which it entered. With line brattice, all rooms have only one single current, and the air, methanous from goaf contacts at first, becomes progressively worse, until at the end it is even more full of methane than it is with auxiliary ventilation. This air is termed "intake air" until it has passed all the rooms operating from the heading, though it is loaded dangerously with methane.

True, passing of air past fan motors may be hazardous, but the cutting machines, loading machines, drills, conveyors and shooting circuits have more dangers, and, being also more numerous and being located at the face, afford greater hazards. At night, if the fans are stopped the air will travel up to the last crosscut, allowing methane to gather only in the remote room ends.

Statistical record of accidents due to explosions in entries where auxiliary fans are in use should be compared with that of entries where air is conducted by line brattice, to see which is the more hazardous practice.

In a mine requiring four-heading entries, the best way of eliminating crosscuts between permanent intakes and returns is not by using an auxiliary fan for ventilation, which has its dangers, but by driving headings in pairs, each pair consisting of an intake and a return and each pair unconnected with the other pairs. When a suitable time comes one pair of headings may be made an intake and the other pair a return (Figs. 3 and 4).

J. W. Woomer,* Wheeling, W. Va.—I should like to speak about bituminous mines and as an engineer in giving my opinion. I believe engineers should be denied the right to revive what should have been dead many years ago. The muddying of the production cycle waters by burdening the foreman, who already is carrying heavy face responsibility, with auxiliary ventilation is nonsense. There is no substitute for correct design of mines and their basic fans.

R. Mancha (author's reply).—I want to bring out something that I do not think was clearly understood; that is, that permissible blowers are already available, and have been available for a number of years; so it is not a matter of developing them or anything like that. And it is not a "plug" for selling blowers.

F. L. Quinby,† Fairfield, Conn.—We have been in the business of manufacturing flexible tubing for nearly 25 years. We do not recommend its use in gaseous mines, but we do think it has a definite place when properly used. We have a very carefully written and carefully edited booklet that we urge all mining engineers and mining schools to use. It is valuable, I think, not only for engineers but also for all operating men who handle this tubing in the mines.

L. E. Young.—I wonder whether Mr. Evans will not make a contribution to this discussion?

C. Evans, Jr.,‡ Scranton, Pa.—I will have to speak cautiously, because in our state there are certain rules about the use of tubing which, in times past, have been enforced in such fashion that tubing could not be used at all. Those rules are now enforced in such fashion that tubing can be used when it is safe to use it; and I will therefore say that those who condemn tubing utterly are thinking only of their local underground condition, and are not taking into account the wide varieties of conditions that occur. For example, in the anthracite region of Pennsylvania a large portion of the output comes from second mining, which means that the roofing of chambers and roof rock is broken, so that there is no possibility of maintaining an air current as that was conceived when the rules on ventilation were first set down. Those rules provide for a crosscut every 60 ft. in anthracite. It is perfectly impossible to maintain a crosscut every 60 ft. in territory that was opened 75 years ago. Those conditions must be taken care of as you come to them. It is not possible to build stoppings in those openings; it is a physical impossibility to do so except at a prohibitive cost; and the only recourse, therefore, is the use of tubing. That fact is recognized by our present Department of Mines, and we are allowed to use tubing under reasonable regulations.

Reasonable regulations—what are they? Well, they are a set of rules that we set down in an attempt to have something that could be enforced.

The difficulty with the use of tubing comes from the fact that we cannot get our mine foremen to use it always under the regulations that we set down—and that is where the danger lies. The danger is a personal one, as it is in so many of our mining operations.

So there is a possibility for the use of tubing, safely; but it requires extraordinary vigilance on the part of management, clear up to the top.

Member.—In other words, the human element has quite a bearing on this problem.

J. Pursglove, Jr.—I would like to ask Mr. Evans a question. Do the places mentioned contain much methane?

C. Evans, Jr.—No. Any area that has been opened for 80 years—or 50 or 40 years—obviously cannot contain much gas, particularly when the roof measures are broken. There are no accumulations of gas, or pockets with any heavy concentration.

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* Mining Engineer.
† E. I. du Pont de Nemours and Company.
‡ General Manager, The Hudson Coal Company.

Use of Rock Dust to Prevent Dust Explosions in Coal Mines, 1938–1943

By H. P. Greenwald,* Member A.I.M.E.

(New York Meeting, February 1944)

This paper brings forward a discussion that was prepared for the meeting of the Coal Division in Chicago in 1938.[1] War in Europe less than a year after that meeting, followed by our defense preparations and entry into the conflict, caused general cessation of large-scale experimentation on the subject, and it is best to confine this paper to events in the United States during the five-year period since the preceding paper was written. The views of members of the Coal Division's Committee[2] on Rock-dusting for 1943 were obtained before this paper was written and the manuscript was reviewed by them.

Extent of Rock-dusting

Statistics on the use of rock dust in the United States during the years 1930–1933, appearing on page 97 of Volume 139 of the Transactions are extended in Table 1 herein through the year 1941.

The results through 1933 did not appear encouraging, but there has been improvement since that date. It has always been true that the larger operations were the ones that used rock dust. In 1941 only 10 per cent of the mines in the country used rock dust, but they produced 57 per cent of the coal, employed just under 50 per cent of the miners, and accounted for 52 per cent of the man-hours. Unfortunately, collection and compilation of these statistics are slow procedures, and figures dealing with the tremendous war expansion of the industry are not likely to be available for a year or more.

Table 1.—*Extent of Rock-dusting in the United States*[a]

Year	Percentage of all Underground Mines Using Rock Dust	Percentage of Total Underground Production from Mines Using Rock Dust	Percentage of Total Underground Employees in Mines Using Rock Dust	Percentage of Total Underground Man-hours Worked in Mines Using Rock Dust
1930	8.4	33.2	27.5	30.1
1931	8.3	31.9	26.8	29.1
1932	7.5	30.6	25.7	27.1
1933	7.2	30.7	25.4	27.4
1934	6.8	33.2	28.3	29.8
1935	7.0	34.2	29.0	30.7
1936	7.4	39.3	32.5	35.7
1937	8.5	42.7	36.3	39.2
1938	7.2	39.7	33.2	35.8
1939	7.9	48.7	40.0	42.7
1940	9.0	55.5	46.4	50.6
1941	10.2	57.1	49.6	52.0

[a] These figures are abstracted from a Bureau of Mines mimeographed report, Health and Safety Statistics No. 323, dated Aug. 21, 1943.

In 1941 the five leading states in regard to percentage of man-hours worked in mines using rock dust were: Utah, 92.1; Wyoming, 79.3; Alabama, 77.4; New Mexico, 72.5; West Virginia, 65.3. The first four states were the same as in 1933, but West Virginia displaced Washington in fifth place.

On the basis of pounds of rock dust used per ton of coal, the five leading states in

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* Supervising Engineer, Experimental Coal Mine Section, Bureau of Mines, Pittsburgh, Pa.

[1] H. P. Greenwald: Recent Trends in Rock-dusting to Prevent Dust Explosions in Coal Mines. *Trans.* A.I.M.E. (1940) **139**, 87–108.

[2] The committee as appointed was: H. P. Greenwald, *Chairman;* C. E. Berner, Daniel Harrington, John E. Jones, E. J. McCrossin and L. E. Young. The death of Mr. McCrossin on Aug. 29, 1943, was a grievous loss.

1941 were: Alabama, 3.32; New Mexico, 2.17; West Virginia, 1.76; Colorado, 1.38; Utah, 1.22. Since 1933 Alabama had moved from fourth to first place and West Virginia from eighth to third, forcing Pennsylvania down to sixth. The average for the United States was 1.31 in 1941 as compared with 0.76 in 1933.

TABLE 2.—*Summary of Findings of Federal Coal-mine Inspectors in District A*

Line No.	State	Pennsylvania					Ohio		West Virginia	All
	Coal Bed	Pittsburgh	Upper Freeport	Lower Freeport	Upper Kittanning	Lower Kittanning	Pittsburgh	Middle Kittanning	Pittsburgh	Total or average
1	Number mines considered	94	41	19	15	47	31	32	53	332
2	Number 100 per cent hand loading into cars	54	25	9	9	32	18	16	17	180
3	Number 0 per cent hand loading into cars	26	7	4	2	9	10	12	23	93
4	Number using rock dust in main haulage	47	19	6	5	22	14	2	31	146
5	Number using rock dust in secondary haulage	43	18	6	3	18	13	2	26	86
6	Number using rock dust in back entries	34	10	3	0	9	4	1	7	68
7	Number using rock dust in rooms	34	15	3	2	5	2	1	20	82
8	Number using no rock dust	46	22	13	10	25	17	29	18	180
9	Number damp to wet mines	18	3	0	2	5	1	7	2	38
10	Number damp to wet mines using rock dust	1	1	0	1	1	0	1	2	7
11	Number completely rock-dusted (see text)	22	8	3	0	5	0	1	6	45
12	Number completely and adequately rock-dusted (see text)	13	4	2	0	2	0	1	2	24
13	Percentage of all mines using no rock dust	49	54	68	67	53	55	91	34	54
14	Percentage dry mines using no rock dust	38	53	68	69	50	53	92	35	51
15	Percentage of line 2 mines using no rock dust	54	60	78	100	57	72	100	59	65
16	Percentage of line 3 mines using no rock dust	50	14	75	50	33	40	83	22	57

FEDERAL COAL-MINE INSPECTION REPORTS

The terms of the law establishing Federal inspection of coal mines and the operations conducted thereunder should now be sufficiently familiar to coal-mining men to obviate the necessity of giving details here. There is accumulating in the files of the Bureau of Mines a mass of information on general coal-mine safety practices, including rock-dusting. Daniel Harrington, chief of the Bureau's Health and Safety Service, kindly arranged that two mining engineers review 870 reports made between December 1941 and March 1943. These covered the eight districts into which the United States is divided under the Bureau's scheme. The writer has found time to make a detailed analysis of the review for one district only—district A, which comprises Ohio, northern West Virginia, Virginia, Maryland, and Pennsylvania bituminous. The review covered 390 reports of inspections made in this area. The mines inspected worked 17 different coal beds, but in only 8 instances were there 10 or more mines per bed in a given state; less than this number is considered to be too few to justify analysis. The figures of the analysis appear in Table 2. Remember that practically all these mines came within the first year's inspection activity and that in general the larger mines received first attention. It is reasonable to assume that on the whole better than average practices would be found in a majority of cases.

In the review of inspection reports, the engineers reported the extent to which hand loading into cars and other methods of loading were practiced. The term "mechanization" is avoided here because under the system used hand loading onto conveyors would have to be called mechanical loading, and on this there will not be agreement. Mines not included in lines 2 and 3 of Table 2 were in various stages between 100 and 0 per cent hand loading into cars. The percentages based on total mines are: hand loading into cars, 54; no hand loading into cars, 28; all others, 18.

Lines 4 to 12 of Table 2 give summary figures from which, with other data, the percentages appearing in lines 13 to 16 are computed. Line 13 shows that, on a general average, more than half the mines inspected used no rock dust whatever. The only individual figure much better than the general average is the Pittsburgh bed in West Virginia. In line 14 allowance is made for wet mines, but the percentages are little changed except for the Pittsburgh bed in Pennsylvania, for which there is some improvement. Comparison of lines 15 and 16 shows that in some individual beds and states the mines using no hand loading into cars make a much better showing than those at the opposite extreme; a less marked difference of the same character appears in the general average.

Finally, it is necessary to point to the figures in lines 11 and 12 of Table 2, which are in some ways the most important of all. Complete rock-dusting is defined as use of rock dust in all areas where the standards demand it. Adequate rock-dusting is use of a sufficient quantity to produce the degree of protection intended. Thus rock-dusting can be complete but not adequate, and may be "adequate where done" without being complete. The comment in quotation marks was met frequently in the inspection reports.

Note in this connection that the inspectors follow the Bureau of Mines standard as set forth in Mine Safety Board decision 32,[3] which requires rock-dusting to within 40 ft. of all faces, minimum 65 per cent incombustible and 1 per cent additional for each 0.1 per cent gas in the air current. This is more severe than the Recommended American Practice approved by the American Standards Committee. Revision of this Recommended Practice was a duty of this committee in past years, but it was concluded that there was not sufficient evidence to support fully any change that might be proposed. The matter cannot become active again until after the close of the war.

There were only 24 completely and adequately rock-dusted mines (line 12); 21 in Pennsylvania, and 13 in the Pittsburgh bed in that state. The total number (24) is 7 per cent of all mines considered and 16 per cent of the number using rock dust. It should be remembered also that rock-dusting was only one of many hazards concerning which the inspector sought information, and the proportion of each report devoted to rock-dusting is small. If a thorough explosion-hazard investigation were made of each of these 24 mines, it seems quite possible that some of them would have difficulty in remaining in the class to which they have been assigned in Table 2.

PART PLAYED BY ROCK DUST IN MINE EXPLOSIONS

The Bureau of Mines file of reports of coal-mine explosions was examined for the five-year period ended June 30, 1943, and a compilation made of all in which dust participated. Purely local gas explosions were not counted, although a large number occurred, with a lengthy toll of death and injury. There were 57 explosions involving dust in the five-year period; in 32 of these no rock dust had been used in the part of the mine traversed by flame, leaving 25 in which rock dust may have played a part.

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[3] Mine Safety Board Decision 32, Prevention of Coal-dust Explosions by Rock Dusting. U. S. Bur. Mines *Inf. Circ.* 7109 (March 1940) 3 pp.

In the 32 explosions in areas that had not been rock-dusted there were 344 deaths and 95 nonfatal injuries, and more than 1242 escaped from the mines unharmed. The exact number of escapes is uncertain, as in three cases the reports do not specify exactly the number of men in the mine. In four of the explosions the number of deaths ranged from 28 to 74, with a total 237. The remaining 28 explosions killed 107 men. In two of the four large explosions, scarcely anyone escaped death or injury. In the remaining two there were a large number of escapes; 134 in one instance and 287 in the other. Both of the latter explosions were of limited extent, and considerable areas of the mines were not affected. Although the figures on the number escaping are interesting, conclusions should be drawn therefrom with caution, as conditions varied widely. In some strong explosions there were few men in the mine, and none escaped death or injury. In seven weak explosions covering only limited areas, 1014 men escaped unharmed. Finally, and this must be emphasized, 24 of the 32 explosions were started by ignition of gas. Had there been adequate ventilation, these 24 would never have started. The eight explosions not started by ignition of gas caused 105 deaths and 41 nonfatal injuries.

The 25 explosions in which rock dust played some part caused 287 deaths and 44 nonfatal injuries, and more than 2153 persons escaped unharmed. Whatever may be the significance, it is a fact that twice as many escaped per fatality and injury in these explosions as did so in the 32 explosions in which no rock dust was present. The 25 explosions fall into two groups: (1) those in which there was rock-dusting close to the origin, and (2) those in which there was not. Nine explosions in the latter group caused 255 of the 287 deaths and allowed only 547 escapes. There were several clear-cut examples of the instantaneous effect of rock dust close to the origin of an explosion. In two instances there were strong electrical ignitions on haulage roads, which were snuffed out at once, with death and injury only to those close to the point of accident. In two others, rock-dusting had been carried close to the face, and the flame was killed within a remarkably short distance—30 ft. in one case. In one instance rock dust in paper bags suspended near the roof takes credit for stopping a well-developed explosion. Finally, and with emphasis again, 21 of the 25 explosions started from ignitions of gas. The other four caused only six fatalities and eight nonfatal injuries and permitted 112 escapes. It so happens that in these four cases few men were in the mines involved. Note also that the four explosions not involving gas caused an average of 1.5 deaths and 2 nonfatal injuries per explosion, whereas the eight similar ones with no rock dust present caused an average of 13 deaths and 5 nonfatal injuries per explosion.

The importance of having adequate rock-dusting throughout is emphasized by comparison of two examples. In one dry and dusty mine a great deal of rock dust had been used but despite this the percentage incombustible in the dust was generally too low. A moderately strong explosion developed, traveled through about half of the mine, manifested force to the surface, and killed two thirds of the entire underground personnel. Contrast this with an incipient explosion initiated near the bottom of a slope by a trolley wire brought down by derailed cars. The area was well rock-dusted. Flame covered an area about 300 ft. long and caused nonfatal burns to a trip rider who happened to be in its path. The motorman at the opposite and inby end of a 27-car trip (the trip was being pushed to the slope, and the first two cars were derailed) was asphyxiated, but not burned. Fifty-one men escaped from workings inby the explosion, led by the foreman, who piloted them around most of the small area contaminated by afterdamp. Had there been no rock-dusting in the immediate vicinity of the accident probably all would have perished.

SUMMARY

1. Statistics show steady but slow increase between 1934 and 1941 in the number of mines using rock dust, the percentage of underground employees working in such mines, in total underground man-hours worked, and in number of pounds of rock dust used per ton of coal produced.

2. The first year's activity of Federal mine inspectors in the territory comprising Bureau of Mines district A showed that more than half the mines inspected used no rock dust at all and that only 24 of 332 mines could be classed as completely and adequately rock-dusted. It is possible that thorough explosion-hazard investigations would reduce this number.

3. Reports of 57 dust explosions in mines over the five-year period ended June 30, 1943, showed that in 32 no rock dust had been used in the area traversed by flame; 344 deaths and 95 nonfatal injuries were caused by these 32 explosions. In nine other places there had been without question incomplete and inadequate rock-dusting, and in these 9 there were 287 deaths. In the remaining 16 instances, rock dust played an important part in stopping the explosion; in four, it was apparently the sole cause of stoppage. These 16 explosions caused 32 deaths.

4. If the present rate of progress is not improved, it seems likely that half a century will elapse before dust explosions are eliminated from all American coal mines. Back entries and rooms are the weakest spots in the present picture of generalized rock-dusting, but they are weakest purely by comparison because the whole picture is unsatisfactory. Simply using more rock dust is not the complete answer. Control of production and scattering of coal dust and major improvements in ventilation practices must proceed at the same time. Following the war there should be conducted a thorough experimental study of the effectiveness of rock dust under conditions closely simulating those in producing mines. It is a fact that the data obtained in some 1800 explosion tests in the Experimental coal mine have put the fundamentals of the problem beyond question, and it is equally true that the information obtained has been honored more by neglect than by use. However, the time has now come to consider special cases and obtain information on them.

DISCUSSION

(*Richard Maize presiding*)

J. V. McKenna,* Waynesburg, Pa.—I want to emphasize certain pertinent points brought out in the paper relative to the areas affected in size and intensity by explosions in mines that were rock-dusted and by contrast in mines that were not rock-dusted. After an exhaustive study by the United States Bureau of Mines and the various state mining departments, it has been definitely determined that an explosion will propagate and travel for great distances unless there is some noncombustible agent present or introduced to stop or hinder the progress.

Various attempts were made to stop the propagation of explosions by the use of water, but this was generally confined to haulageways and usually was not satisfactory because an insufficient amount of water was used and because the water evaporated quickly.

No attempt was made at that time to allay the coal dust at its point of origin and it was carried by the air currents to almost any part of the mine.

Finally rock dust was introduced, but its adoption was a slow process, as stated by Mr. Greenwald. Some states have passed laws making it mandatory that gaseous and/or dusty mines be rock-dusted. The Pennsylvania

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* Inspector, Pennsylvania Department of Mines.

Bituminous Mine Law directs that: "Rock-dusting shall be done with such regularity and frequency that all surfaces required to be rock-dusted shall be kept in such a condition that the incombustible content of the adhering and lodging dust shall not be less than 55 per cent." However, we try to keep the incombustible content at 65 per cent.

"The rock dust shall be distributed on top, bottom, and sides of all haulageways, traveling ways, airways, developing entries, and entries producing coal from rooms or pillars and their parallel entries." The superintendent shall see that a representative sample of dust is gathered at each sampling point once each two months and tested to determine the percentage of incombustible material. The distance between sampling points on haulageways, traveling ways, and airways shall not exceed 2000 ft., but in developing entries and in entries producing coal from rooms or pillars, and their parallel entries, the distance between sampling points shall not exceed 500 ft. A record of all samples taken must be kept in a state record book provided for that purpose.

When rock dust was first introduced it was generally applied by hand, but this was found to be a slow and expensive method and was supplanted by the mechanical method of application.

A survey made of the gaseous mines in the Thirteenth Bituminous District in Pennsylvania reveals that the average amount of rock dust used in the year 1943 was 1.71 lb. per ton of coal mined. The relation of pounds of rock dust used to man-hours of exposure reveals that approximately 85 per cent of the coal mined in the district, which also is practically the same percentage of man-hours exposure, was rock-dusted. This survey covers 19 mines, of which 10 used rock dust.

To get the desired results, the first application of rock dust usually takes about 4 lb. of rock dust to the linear foot and the second application, or redusting, takes from 1½ to 2 lb. per linear foot when the dusting is done mechanically. When the dusting is done by hand considerably more rock dust is required—as much as 10 to 12 lb. per linear foot on occasions—because the dust does not stick to the roof and sides and falls to the bottom. In many cases of hand rock-dusting that I have observed, the bottom would have most of the dust and the roof and sides would be streaked and not dusted evenly.

When the mines were working one shift and only five days a week the matter of application of rock dust was comparatively simple, as the rock-dusting could be done on the off shift or on the off day. However, since practically all mines in western Pennsylvania have gone to multiple shifts, the problem of rock-dusting has become exceedingly difficult, particularly in mechanically operated mines working three shifts and six days per week. About the only time rock-dusting can be done in this case is on Sunday and it is difficult to get men to work on Sundays, even though they may have missed a shift through the week. If a man works every day through the week and then works on Sunday, it entails double time, thereby increasing the cost considerably. If certain necessary maintenance crews were allowed to start a work week on Tuesdays, it would simplify considerably such jobs as rock-dusting, necessary timbering or track work. Rock-dusting, particularly, must be done when no work is going on in the section being rock-dusted, as visibility would be zero and any person on the return side of that split of air would practically be suffocated.

In Mr. Greenwald's paper is the statement: "Complete rock-dusting is defined as the use of rock dust in all areas where the standards demand it. Adequate rock-dusting is the use of a sufficient quantity to produce the degree of protection intended. Thus rock-dusting can be complete, but not adequate and may be 'adequate where done' without being complete." The statement "adequate where done," it is said was met frequently in Federal Mine Inspection Reports. I am afraid that this statement applies to many of our mechanical mines in southwestern Pennsylvania because we do not attempt to rock-dust within 40 ft. of the face, as this is almost impossible in a three-shift mine.

To go back to the rock-dusting that must be done when the sections are idle, which is only on Sunday. In the meantime the place may have traveled 100 ft. However, in the mines I speak of in this discussion, we have what we feel is a better protection against propagation of explosions than the use of rock dust only. In all the mechanical mines referred to here, there are sprinkling lines in all working places.

A hose is attached to the end of the line and then connected to the cutting machine, and the dust is sprayed continually during cutting. The coal is again sprayed before shooting and also during the loading operation, by the loading machine, in the same manner as in the cutting operation.

In some places it may not be necessary to sprinkle when the actual loading is in progress, if the coal has been sufficiently wetted when cutting operations and shooting are done. This, of course, is a matter of judgment, as conditions in mines vary greatly. Therefore we do feel that we are giving better protection by using water at the face in conjunction with the use of rock dust rather than by using rock dust alone and keeping it within 40 ft. of the face. We feel that we are knocking the coal dust down at its source and it cannot then be carried to any other parts of the mine. Holes are drilled through the bottoms of the cars, which serve two purposes—they let the excess water drain out of the mine cars and they tend to keep a part of the haulageway damp enough to allay the coal dust. All development working areas in these mines are rock-dusted weekly and the retreat portions as needed. The mines are 100 per cent rock-dusted as far as that percentage can be reached. The working areas are rock-dusted by low-pressure rock-dusting machines usually. The back entries and air courses are rock-dusted by high-pressure machines, which is a much more difficult task, as a considerable amount of hose must be used to reach some of the entries. A pipe big enough to permit the passage of the rock-dust hose is placed in strategic stoppings and a cap is screwed over the pipe.

In mines where hand loading is practiced, the problem of rock-dusting is not quite as difficult because the working places do not advance as rapidly, and the rock dust can be more easily kept within 40 ft. of the face.

There are other places in the mines where the use of rock dust would not be feasible even though there is considerable coal dust thrown into suspension, particularly at underground dumping points. This dust must be knocked down, and the only way to do it satisfactorily is with water. However, the amount of water used at these places can be reduced materially by the use of a good wetting agent. The dust must be kept down at the dumping points, and sometimes the use of water alone makes the coal so wet that the moisture content is too high, and it interferes with the transportation of the coal at times on a belt system; therefore the use of the wetting agent to cut down the amount of water used.

I know of one place where a large galvanized pipe has been installed at an underground dump. The end of the pipe was placed at the point where the greatest amount of dust accumulated in the air, and carried from this point to the return air course. The dust was carried through this pipe to the return air course and treated with rock dust at that point. This arrangement reduced the amount of water that had to be used on the dump to keep the dust under control.

I want to emphasize a statement in Mr. Greenwald's paper, that adequate ventilation will stop the ignition of gas, which in turn is fed by coal dust. Under proper supervision, and if a sufficient amount of air is properly coursed and locked to ensure a continuous flow of air, there normally should be no accumulations of explosive gas. However, rock dust must be used as a second line of defense; and as a first line of defense in case of a sudden inrush of gas from any source.

The possibility of a gas ignition from a sudden inrush of gas is lessened considerably if the air currents are not carried over trolley wires after they have ventilated working places. Many explosions have been ignited by a spark from a trolley. This arrangement of ventilation can be accomplished easily by the use of multiple entries, with enough fresh air going over the trolley wire to keep it clear.

In regard to explosions from coal dust alone, adequate use of rock dust is still the answer, and is the only line of defense that can be generally used in coal mines.

E. McAuliffe,* Omaha, Nebr.—Rock-dusting is one of the things that too often go by default. I can recall when we started rock-dusting in Wyoming. One of my neighbors came over and bought a quarter of a ton from us and stored the sacks outside his mine, and thereafter he was reported for some years in one of the labor journals as operating a rock-dusted mine. It is that attitude of mind, I think, that has cost a great many lives and held up safety.

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* President, The Union Pacific Coal Company.

While we attempt to rock-dust our mines, we have not found it practical, as Mr. McKenna said, to rock-dust right up to the face. We do get a continuing pressure from federal inspectors to bring our rock-dust closer to the working face. We are now working 16 hr. a day, and working one shift, and with the hope of working two shifts on Sunday there is a great deal of difficulty attached to the application of dust. Nevertheless, we are going to carry on our rock-dusting if we have to shut down the mines for a shift, with a resultant loss in production.

I am a definite and firm believer in ventilation of the mines, as Mr. McKenna properly said. That is the real first line of defense and one that is too often neglected. Given adequate ventilation—and it is sometimes difficult to get the local force to maintain full ventilation —a little bit of prodding from a federal or a state inspector once in a while is not altogether a bad thing.

The next thing is to allay the dust where it is produced. We undertook to put water on our cutter bars in all our working places some years ago, finding it necessary to install several miles of pipe line. We have no such mileage of water lines at the present time because we were then operating 19 widely scattered, hand-loaded mines. Concentration of the working area through mechanical loading has reduced that mileage to relatively insignificant amount, but we do keep water on the cutter bar of every machine, and even under the present difficult labor conditions no man can stay with us that persistently attempts to cut without water.

After the mining machine is retracted and the crew gets ready to load, the unit foreman determines the measure of water that should be put on the cuttings and on the rib and floor —not so much as to wet the coal so that it cannot be moved successfully in a rather rigorous Wyoming winter.

Those are things that can be worked out, and it is astonishing how easily a good practice can be maintained once established, providing the management has the will to establish it and see it through.

S. A. Taylor,* Pittsburgh, Pa.—I want to give the result of a little experiment that was made long before the Mining Bureau was established. I was still in college. Dr. Francis Phillips was chief of the Chemical Department at the University of Pittsburgh. (Then it was the Western University of Pennsylvania.) Natural gas had just been discovered in Murraysville, Pa., and of course it went to waste there for a number of years. Finally George Westinghouse took hold of the proposition and he employed Francis Phillips to make an investigation as to how it could be used. Dr. Phillips made a very thorough investigation and chemical analyses and at that time his investigations were more complete than any that had been made in any other place in the world, even Germany, where he had gone to school.

One day, I think it was in 1886, he took our class into his laboratory to show us the effect of exploding natural gas. He had demonstrated that the explosibility of natural gas had limits on the low side of five and on the high side of 15, which corresponds pretty closely to what the Bureau of Mines established after a great many experiments. He made an experiment there to show us how dangerous or how powerful an explosion would be. He used a little water glass, a drinking glass. He fused a glass tube into the side of the glass, which was connected with a natural-gas tank and also with air. He had electrodes inserted in the glass, so as to explode the mixture. He put that glass over a vacuum pump and created a vacuum, after which he admitted 10 parts of natural gas, because it was at that point he figured would be the most powerful explosion, and added 90 parts of air. He fired that mixture with the electrodes, and the explosion from that small quantity of gas and air mixture was so powerful that we could feel the vibration of the building, which was a three-story brick structure.

I think the taking care of gas is one of the most important things in mining. I know this knowledge has served me very well in my own experience, and it is a practical demonstration of the powerfulness of gas. To keep gas out of the mine is one of the things I think is most important.

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* Consulting Engineer.

Standards for Safety Clothing, and Their Relation to Accident Reduction

By C. E. Berner,* Member A.I.M.E.

(New York Meeting, February 1944)

Since 1933, the Coal Mine Rating Schedule in Pennsylvania has provided credits in the Workmen's Compensation insurance premium rates for the use of safety hats, goggles and shoes. In that year, the worth of such protective clothing had just been well established and satisfactory articles were being made by a number of manufacturers. Standards for these three items of safety clothing were written in the Rating Schedule and have been little changed through the years.

The credit in the premium rate for a bituminous mine is $0.14 for hats, $0.08 for goggles, $0.08 for shoes. In order to earn such credit, it is necessary that all the employees in the particular mine be equipped with safety hats, goggles or shoes, as the case may be. In fact, it is not unusual for mines to qualify in all three categories, thus earning a rate reduction of $0.30 per $100 of payroll.

These credits have been a decided stimulus to the use of safety clothing, and at present insured bituminous mines are credited as follows:

Hats........	63 per cent, or 22,600 employees
Goggles.....	28 per cent, or 10,000 employees
Shoes.......	44 per cent, or 15,800 employees

Of course, many other employees wear these protective articles but they are in mines that are not fully equipped and so are not given the rate credit.

The growth in use of this equipment has not been spectacular but it has been steady, as shown in Table 1. An interesting thing about this is that it covers virtually all of the small and medium-sized bituminous mine operations in Pennsylvania. A number of large operations are included, but many large mines are self-insured.

Table 1.—*Number of Mines Given Credit*

Year	1934	1935	1936	1937	1938	1939	1940	1941	1942
All mines inspected	412	438	429	627	526	514	512	563	564
Credit: Hats	28	46	45	88	125	181	223	263	295
Goggles	21	25	25	36	57	72	83	99	106
Shoes	25	42	42	72	87	118	141	169	170

It is difficult to keep alive interest in safety work in the smaller mines: first, because employment of safety engineers is out of the question and the one or several mine officials employed are often concerned only with production; second, because the life of many small mines is brief, so that safe conditions although established do not last long.

Conditions as to safety in mines of this class rest chiefly with inspectors of state mine departments and of casualty insurance companies. Many millions of tons of production come from these mines, so that

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they are of real importance, and their safety should not be neglected.

I believe the modern progressive operator is thoroughly alive to the value of every possible protection that can be afforded employees and is convinced of the need of protective clothing. However, as with many other excellent devices, their use is not standardized or uniform and many accidents occur from day to day that could have been prevented through the proper provision and use of some safety apparel. I am led to this conclusion by a study of injuries in Pennsylvania during recent years, as shown in Tables 2 to 5.

TABLE 2.—*Labor and Industry Records in 1941 for all Coal-mining Temporary Disability Cases*

Part of Body (1)	Anthracite Mining			Bituminous Mining		
	Accidents		Compensation (4)	Accidents		Compensation (7)
	Number (2)	Per Cent (3)		Number (5)	Per Cent (6)	
Total	5,433	100.0	$755,032	7,223	100.0	$1,207,432
Eyes	256	4.7	13,906	254	3.5	13,511
Head	311	5.7	43,727	366	5.1	60,099
Chest	535	9.9	76,942	679	9.4	83,740
Back	656	12.1	134,508	952	13.2	266,730
Arms	236	4.3	36,841	285	3.9	38,798
Hands	461	8.5	31,420	488	6.8	50,064
Fingers	944	17.4	54,817	1,036	14.3	67,022
Legs	827	15.2	171,724	1,140	15.8	279,540
Feet	612	11.2	87,094	992	13.7	153,791
Toes	260	4.8	24,342	355	4.9	28,984
Abdomen (incl. hernia)	47	.9	13,241	239	3.3	44,185
Body (N.O.C.)	288	5.3	66,470	437	6.1	120,968

TABLE 3.—*Temporary Cases in All Pennsylvania Industry in 1941, by Nature of Injury*[a]

Part of Body (1)	Number (2)	Amputations (3)	Burns and Scalds (4)	Concussions (5)	Cuts and Bruises (6)	Fractures and Crushes (7)	Hernia (8)	Infection (9)	Sprains and Dislocations (10)	Unclassified (11)
Total	54,252	194	2,596	315	23,992	11,630	1,737	3,382	9,653	753
Eyes	1,688		317		1,277			86	2	6
Head	3,072		417	310	1,848	327		67	64	39
Chest	4,121		47		1,563	1,550		17	835	109
Back	5,796		40	3	1,139	364		19	4,167	64
Arms	2,964		255	1	1,349	802		213	320	24
Hands	5,483		478		2,599	1,056		756	555	39
Fingers	9,069	193	111		5,336	1,899		1,387	108	35
Legs	6,741		348	1	3,633	1,214		351	1,145	49
Feet	7,262		409		2,871	2,251		205	1,473	53
Toes	3,080	1	20		1,049	1,913		63	19	15
Abdomen	1,737						1,737			
Body (N.O.C.)	3,239		154		1,328	254		218	965	320

[a] Pennsylvania Department Labor and Industry, 1941 Statistical Supplement, Industrial Injuries in Pennsylvania.

A careful review of the foregoing facts indicates that there is certainly great room for improvement to be effected by complete use of protective clothing. It is my assumption that the situation in many other coal-mining states is the same as in Pennsylvania. In fact, it is my belief that most states are in a less favorable situation than Pennsylvania and especially so at this time. I am led to this observation by the figures released by the Pennsylvania

TABLE 4.—*Insurance Experience in Pennsylvania*, 1936–1940
COMPENSABLE ACCIDENTS IN RELATION TO SAFETY CLOTHING[a]

Injury	Anthracite Mining				Bituminous Mining			
	Compensable Accidents	Days Lost	Percentage of Compensable Accidents	Percentage of Days Lost	Compensable Accidents	Days Lost	Percentage of Compensable Accidents	Percentage of Days Lost
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)
Total	1,597	310,253	26.9	14.2	4,393	611,160	25.8	11.5
Fractured skull	38	174,246	0.6	8.0	73	239,686	0.4	4.5
Cut or bruised scalp	145	7,278	2.4	0.3	246	17,223	1.4	0.3
Cut eye	257	74,713	4.3	3.4	593	144,585	3.5	2.7
Cut palm of hand	83	3,176	1.4	0.2	250	26,935	1.5	0.5
Cut thumb or finger	510	17,971	8.6	0.8	1,429	57,894	8.4	1.1
Fractured foot	79	7,991	1.3	0.4	364	60,668	2.1	1.1
Cut or bruised foot	258	10,584	4.4	0.5	742	31,413	4.4	0.6
Fractured toes	122	11,680	2.1	0.5	400	24,554	2.4	0.5
Cut or bruised toes	105	2,614	1.8	0.1	296	8,202	1.7	0.2
All compensable accidents	5,943				17,012			
Total days lost	2,177,721				5,310,037			
Production, net tons	23,730,024				137,202,918			

[a] This is not all the injuries of this kind but only the ones that upon examination seemed preventable by the use of safety clothing.

TABLE 5.—*Noncompensable Accidents by Nature of Injury and Part of Body—Insured Mines in Pennsylvania*, 1940–1942[a]

Part of Body	Anthracite						Bituminous					
	All	Percentage of All	Cut or Bruise	Sprain or Dislocation	Burn	Fracture	All	Percentage of All	Cut or Bruise	Sprain or Dislocation	Burn	Fracture
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)	(11)	(12)	(13)
Total	5,612	100.0	4,615	882	82	33	11,409	100.0	9,131	1,744	380	154
Eye	1,035	18.4	1,019		16		1,788	15.7	1,697		91	
Face or scalp	849	15.1	833		16		1,047	9.2	1,005		42	
Back	583	10.4	128	442	13		999	8.7	217	750	32	
Chest or groin	325	5.8	210	105		10	733	6.4	424	233	11	65
Shoulder	179	3.2	94	82	2	1	359	3.1	208	147	4	
Upper arm	26	0.5	21	5			75	0.7	61	12	2	
Elbow	103	1.8	96	7			162	1.4	151	11		
Lower arm	108	1.9	104	4			209	1.8	177	19	13	
Wrist	186	3.3	139	45	2		314	2.7	171	117	26	
Hand	318	5.7	308	3	7		645	5.6	567	15	60	3
Thumb and finger	806	14.4	775	11	10	10	2,326	20.4	2,219	26	31	50
Upper leg	120	2.1	92	18	10		385	3.4	291	43	51	
Knee	246	4.4	173	73			566	5.0	444	119	3	
Lower leg	111	2.0	103	7	1		329	2.9	306	18	5	
Ankle	106	1.9	47	58	1		304	2.7	125	175	4	
Foot	388	6.9	364	19	3	2	852	7.5	786	56	4	6
Toe	123	2.2	109	3	1	10	316	2.8	282	3	1	30

[a] The accidents listed here are of 7 days duration or less and include those having medical cost and no lost time.

This table represents only part of the noncompensables, as there were 9304 such cases in insured anthracite mining, and 17,276 in insured bituminous mining. The distribution in this table should hold good for the foregoing totals. It is of interest because it is one of the few analyses made of any large number of noncompensable injuries in coal mining.

Department of Mines for the first eight months of 1943. The fatality rate for bituminous mining is reported as 1.32 per million tons or "36 per cent better than the coal industry of the nation." Another recent Pennsylvania report states "there has been a steady decline for almost six months in the number of accidents reported to the Labor and Industry Department"—this in spite of continued high employment in the state.

Survey of Present Situation

In looking over available material on this subject, I learned that in 1933 a very thorough and excellent presentation of this subject was made by Currie and Fene, of the United States Bureau of Mines. At first it seemed that anything I might say today would be more or less a repetition of what was said 10 years ago. However, it did seem desirable to attempt a thorough analysis at this time, in order to note what progress, if any, had been made in the past decade. This is especially true because safety clothing for mine employees was little used until 15 or 20 years ago. In addition to the paper by Currie and Fene, some excellent articles have appeared in the magazine *Mechanization*, during the year 1942. Some of this material was based upon records of the Coal Mine Section.

In an effort to learn more accurately the place of safety clothing in the coal-mining industry, we sent letters to many of the larger operators in all coal fields. Replies were received from companies representing production of approximately 120,000,000 tons. The questions asked and composite replies were as follows:

1. What percentage of your employees wear hard hats, safety goggles and safety shoes?

Most of the operators report 100 per cent use of hats and shoes. Most of the operators report that men on machines and certain inside and outside company men are equipped with goggles. A few operators report that all inside employees have goggles.

2. Are safety goggles worn at all times or only during use of certain hand tools? Do you continue to have eye injuries due to neglect in using goggles as required? What is your opinion of and experience with plastic eye shields?

Only one operator reported use of goggles all of the time. Most operators require use in certain jobs that produce flying particles. All mines continue to have eye injuries due to failure to use goggles as required.

Plastic eye shields have been tried at a number of mines on an experimental scale. At a comparatively few mines, they are provided for all employees. It is doubtful whether the employees in general use the shield any more readily than they use goggles.

3. Are dust masks or respirators used in dusty occupations?

All mines are using respirators in certain dusty occupations—some more widely than others.

4. What standards or specifications are followed in purchasing the foregoing items of protective equipment?

Generally speaking, purchases are made from established reputable manufacturers of safety equipment, with no definite emphasis on specifications. Some operators require that equipment be approved by the U. S. Bureau of Mines or meet the requirements of the National Bureau of Standards.

5. Do your employees use gloves, and do you consider them essential to accident prevention?

The use of gloves seems to vary from a small percentage up to 80 or 90 per cent. Some interesting comment on the use of gloves is recorded under Hand and Finger Protection.

6. Could you furnish any statistics showing improvement in accident record from use of foregoing items?

Only a few operators submitted any statistical data pertaining to injuries to part of body for which protection is afforded by safety clothing. Most operators felt definite improvement had been realized. Records for two groups of mines are given in Table 6. This exhibit is very convincing of the value of safety hats and safety shoes in greatly reducing injuries to head and toes. It also shows that eye injuries continue at a high rate, even though the means of prevention is in the employee's possession. Furthermore, it shows a very large percentage of injuries to fingers, which I believe is true in all mines.

I was surprised to learn from two large operators that hard hats were used by only 50 per cent of employees, hard-toe shoes by 25 to 30 per cent and safety goggles were limited to a few company employees. One of these operators very candidly said: "You will see that safety clothing has been so sparsely used in our mines that we have no worth-while statistics on accident reduction. From time to time, we have had isolated examples of toes saved by safety shoes, goggles shattered and hard hats dented, but we continue to have many accidents, the severity of which would be materially reduced if such equipment were worn. . . . In this area, the use of protective clothing has never had the support of the labor organization." This last statement is not too startling, yet it is significant. For some time it has become increasingly difficult for operators to insist upon or compel compliance with desirable safety regulations. To an increasing degree the active support of labor organizations is being enlisted. Within the last year, definite steps have been taken to have labor representation on committees of national organizations concerned with mine safety. It is hoped that the proper interest and influence on the part of labor representatives will serve to restore and enlarge a vital concern by workers for safety in all its phases.

Experience with the Various Articles of Protective Clothing

Head Protection

The need for head protection in the mines has been apparent for many years. It was not uncommon for miners subject to falling particles to pad their old canvas hats, which acted as shock absorbers. It also seems commonly accepted that great impetus to head protection was afforded after the first World War, when it was definitely known that many lives had been saved by the use of the steel helmet in the armies. Hard hats began to appear soon after the war and were being tried out in many places throughout the industry.

It was some time before there was any definite realization as to just what a hat should be able to withstand in impact; also before there was a general realization that a hat could and should be so constructed as to be proof against electric shock, because with the almost universal use of electricity in the mines, contact with exposed wires was rather common.

In the years 1932 and 1933, the Coal Mine Section worked with a number of hat manufacturers for the purpose of drawing up satisfactory standards for protective hats. By that time, a satisfactory material had been evolved that would withstand considerable impact and a fairly heavy electric charge. An important feature that had to be evolved was the proper cushioning inside the hat, which would guarantee that a heavy blow on the head would not be transmitted directly to the skull. Another important construction feature was the elimination of metal through the hat, which would conduct current from the outside to the inside and thus to the wearer. These problems were satisfactorily worked out and a number of manufacturers are making hats that conform to the standards drawn up at that time. These standards appear in the latter part of this paper. Testing of these hats is conducted by an employee of the Coal Mine Section at the laboratories of the Experiment Station of the Bureau of Mines at Pittsburgh, where proper facilities have been provided. The manner of testing for impact is set out in detail in the standards. In testing for electric resistance, a sheet of tin foil is put over the wooden block and another sheet over the crown of the hat and current is then stepped up until the proper voltage is arrived at and connection is made to the foil underneath and on top of the hat. This test is conducted with a dry hat and then the hat is thoroughly wet, in order to simulate the condition that arises when a worker perspires freely.

TABLE 6.—*Disabling and Nondisabling Injuries*

Year	Head		Eye		Finger		Toe	
	Number	Percentage of Total	Number	Percentage of Total	Number	Percentage of Total	Number	Percentage of Total
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)
Company X. 100 Per Cent Hats, Shoes, Goggles about 1930								
1928	420	7.47	464	8.25	1,183	21.02	275	4.89
1930	387	8.23	478	10.17	1,127	23.91	245	5.21
1932	107	7.08	118	7.81	482	31.90	47	3.11
1934	233	8.27	199	7.06	732	25.97	126	4.47
1936	116	2.50	292	6.29	1,180	25.40	166	3.57
1938	45	2.38	142	7.51	492	26.00	80	4.23
1940	108	2.80	301	7.81	815	21.14	129	3.35
1942	126	2.09	559	9.29	1,377	22.89	163	2.71
Company Y. 100 Per Cent Hats 1934; Shoes Increasing Year to Year—90 Per Cent in 1942; Goggles Complete								
1928	123	6.05	211	10.38	573	28.18	140	6.89
1930	91	7.69	142	12.00	287	24.26	65	5.49
1932	55	8.41	73	11.16	168	25.69	45	6.88
1934	29	4.69	80	12.94	170	27.51	33	5.34
1936	6	0.95	160	25.36	134	21.24	28	4.44
1938	4	0.93	67	15.58	151	33.13	15	3.49
1940	9	1.79	77	15.34	160	31.87	15	2.99
1942	6	1.81	85	25.60	89	26.81	10	3.01

The original hats were heavier than hats in use at the present time and were not as well cushioned, and, therefore, not as comfortable. It was some years before miners generally felt that the hat was not burdensome. In fact, there was a great deal of opposition to the use of the hat at various times and places. An interesting example of this opposition is contained in a letter to the Pennsylvania Insurance Commissioner in August 1933 from a prominent attorney in western Pennsylvania:

> Many groups of miners in this district have registered complaints with me in regard to safety hats or caps which coal operators are forcing them to wear during the entire time spent inside the mine. The complaints show that in many cases the wearing of these safety hats or caps has resulted in serious physical injury to the wearer.
>
> I have found no provision in the laws of this Commonwealth which requires miners to submit to the wearing of these safety hats or caps. However, under Item 2 of General Safety Standards contained in the Pennsylvania Bituminous Coal Mine Compensation Rating Schedule 1933, as approved by you, reads as follows:
>
> "(a) Safety Hats or Caps. All inside employees shall wear safety hats or caps approved by the Bureau, during the entire time spent in the Mine."
>
> Will you kindly advise me what recourse the miners have in case they wish to take issue with this requirement?

In reply, this attorney was informed of the great number of accidents of record in the insured bituminous mines in Pennsylvania, which provided the basis for credit in the workmen's compensation insurance rate where proper protection was given. In addition, this was written to him:

> Permit me to point out to you what a fine, humanitarian and noble work you, as counsel for a group of miners, could accomplish with this information by pointing out to them that the inclusion of these items in our Rating Schedule is primarily for their personal safety and protection and that it is to their own advantage to use this safety equipment.
>
> Your letter raises a question in which we are very much interested, namely, that the use of hard hats has caused "serious physical injury." We have no record of such injuries and are very anxious to obtain authentic data on such accidents. Will you cooperate with us by furnishing such record?

We did not receive any information as to injuries caused by the wearing of hard hats. We have reason to believe that the appeal to him had some effect because opposition on the part of the workers seemed to diminish very rapidly.

It is my opinion that the use of approved hard hats has been a most important factor in preventing many serious and fatal accidents. There is hardly an operator who does not know of one or more instances in his own mine or mines in which accidents have occurred and in which the hard hat very definitely prevented either a serious or a fatal injury. A few examples taken at random are the following:

From Utah: A coal driller received a blow on the top of his hard hat from a slab of coal 3 in. thick, weighing about 40 lb., and falling a distance of 12 ft The impact knocked him down, causing slight abrasions of shoulder and thigh and necessitating loss of two days work. The injured believes the hat prevented a fractured skull.

Open-pit mining: Working on a powder hole, a man wearing a hard hat was struck by 22 lb. rock that fell 20 ft. He was knocked down and suffered a very slight head fracture and a broken shoulder. He was back on the job in six weeks. Without safety hat protection, undoubtedly this would have been a fatal accident.

In a paper in 1936,[1] the late John T. Ryan, Sr. estimated the number of hard hats in the coal-mine industry to be 60 to 65 per cent. Today it is estimated that in the anthracite field 90 to 95 per cent of employees use hard hats and throughout the bituminous regions 80 to 90 per cent use them. This estimate is based upon information from manufacturers and a considerable number of large coal-mining companies.

A standard safety hat is expected to last four or five years, on the average. But many hats are in use that have been damaged by blows of various sorts or in which the lining is badly worn. Such hats will not afford the desired protection. Inspections should be made at regular intervals and hats should be kept up to standard.

Eye Protection

Without doubt, the National Society for the Prevention of Blindness has gone into the matter of eye protection in industry more thoroughly than any other group. In a recent issue of *Mechanization*, I find that this Society estimated the number of unnecessary eye injuries in American industry to be over 300,000 per year. Approximately 20 per cent, or 60,000, are compensable, having an estimated cost of $100,000,000.

In the state of Ohio there is recorded a total of 220,785 injuries in 1926. Of this total, 43,038, or 19.5 per cent, were to the eyes. In 1935, the total reported injuries were 164,572, of which 29,536, or 17.9 per cent, were to the eyes.[2] In a general way these figures substantiate the total of 300,000 industrial eye injuries per year. In the mining industry of Ohio, improvement in the record of eye injuries may be noted from the fact that in 1930 there were 800 compensable eye injuries in the production of 22,035,674 tons of coal, while in 1942 there were 757 such injuries in the production of 31,491,420 tons.

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[1] J. T. Ryan: Recent Advances in Mine Safety Practices and Equipment. *Min. and Met.* (March 1937) **18**, 151.

[2] J. W. Beall: Eye Injuries in Ohio Industry.

Records of the Coal Mine Section, P. C. R. and I. Bureau, are shown in Table 7.

TABLE 7.—*Compensable Eye Injuries, Insured Bituminous Mines in Pennsylvania*

Data	1926–1930		1936–1940	
	Number	Days Lost	Number	Days Lost
Loss of eye	206	370,800	70	126,000
Temporary	1,218	36,540	550	16,500
Total	1,424	407,340	620	142,500
Tons production		310,211,600		137,202,918
Compensable accidents per million tons		4.6[a]		4.5[a]
Days lost per million tons		1,313		1,038

[a] Waiting period in compensation cases reduced from 10 to 7 days, Jan. 1, 1928, so these two results are not exactly comparable.

Several permanent total disabilities occured each year, but are not included in Table 7. All were the result of premature blasts. It is possible that goggles would have saved one or both eyes in some of these cases.

These statistics may seem rather sketchy and there is little or no connection between the various groups covered. They do serve to show most decidedly that eye injuries are still occurring in great numbers. Throughout the coal-mining industry, it is doubtful whether any great improvement has been made in preventing eye injuries during the last 10 years. It is undoubtedly true that a great many men have been equipped with goggles but it is likewise true that the majority of these men do not use them all of the time and frequently fail to use them even when engaged in work that entails flying particles. A few mines have been able to induce their men to wear the spectacle type of safety goggles all the time they are underground, and eye injuries have practically disappeared.

At the present time it seems more difficult than ever to enforce the use of goggles. A great many of the larger mines equip their men with goggles and instruct them how to use them but it is the experience in most of these mines that men will keep the goggles in their pockets or on the tops of their hats.

There is no doubt that certain difficulties exist in mines that are not present in surface operations. Temperature, humidity and dustiness may make it virtually impossible to keep goggles dry and clean. However, a number of mines use water and wetting agents and thereby reduce the dustiness at the face to a minimum and make the use of goggles more feasible.

It is quite possible that the operators have exerted all the pressure and influence they can in promoting this safety feature and that better results will come only when miners and their leaders are convinced of the necessity of undergoing some slight inconvenience in wearing goggles in order to derive the much greater benefit of the saving of sight.

Goggles are not always adapted to the wearers, but at some mines a thorough examination is made of employees with the idea of being able to correct their vision where correction is needed. An instance that has been referred to on more than one occasion is that of the Union Pacific Coal Co., which undertook to provide corrective eye protection in the year 1933. Out of 1742 men examined, only 25 per cent had normal vision. Over 28 per cent were found to have major defects. Similar results have no doubt been found since then or even before by other operators, yet it seems the fact that the majority of miners who are fitted with goggles do not first have a thorough eye examination. The importance of this can hardly be overemphasized today, when older men are replacing younger men who have gone out into the Armed Services or into other industry. Many of these older employees have impaired vision and would profit greatly by corrective lenses for use in underground work.

As an indication that the proper use of goggles will prevent eye injuries, the following examples, published in *Mechanization*, seem to offer conclusive proof:

> Company A, in the Rocky Mountain region, adopted goggles Nov. 1, 1935, no one being permitted underground without goggles after that date. Over a 5-year period the company showed a reduction of 75 per cent in the number of eye injuries and an 84 per cent reduction in eye injuries per million man-hours.
>
> Company E, in West Virginia, over a 3-year period, experienced a reduction of 63 per cent in the number of eye injuries and an 80 per cent reduction in eye injuries per million tons. This cut their compensation costs for such injuries 84 per cent.
>
> Company I, Pennsylvania anthracite, adopted goggles on an extensive scale in 1939. From 1938 to 1941 they showed a reduction in the cost of eye injuries of 82 per cent and a reduction of 84 per cent in the cost of eye injuries per ton.

A prominent safety director expressed an interesting observation in relation to eye injuries. He said that some years ago most men on coal had three or four picks, which usually were well pointed and sharpened. The tendency now is to have one or two picks, often with dull and blunt points. It stands to reason that a dull pick will cause more flying particles of coal or rock and thus increase the chance of eye injury very decidedly. A bad practice such as this may offset much of the improvement gained by years of effort to furnish and use eye protection.

Plastic eye shields can be used instead of goggles. Improvement in plastic materials can be looked for which will overcome objections raised as to scratching and discoloring. One thing is certain, these shields do furnish much protection from flying particles that may cause scars and disfigurement. This is particularly true in Pennsylvania anthracite mining, where cuts from flying coal are very common. In insured anthracite mines during the three years 1940 through 1942, there were 5150 compensable injuries having a total incurred compensation of $1,471,409. Of this number, 630 were cases of disfigurement with incurred compensation of $62,273, and the average award was for 6 weeks.

Foot and Toe Protection

Ten years ago safety footwear had just been well standardized and mine employees had not long been afforded this protection. Because of the more rugged type of construction and the rigid toe cap, it is important that safety shoes shall be well fitted to the wearer. At first there was not sufficient variety in sizes, and poor fitting caused discomfort and aroused opposition; but that was corrected as the sales increased. Recently, one large shoe manufacturer said that "the demand for all types of safety shoes has greatly exceeded the supply." Perhaps that is due now to the scarcity of shoes for civilians, but it should be remembered that mine workers can secure orders for the necessary safety footwear.

Most of the large operators report that all employees have safety shoes, but this is far from true in medium and small mines in Pennsylvania and probably in similar mines in all states. Improvement in this respect in this group of mines again devolves upon state and insurance company inspectors.

It would seem that toe accidents are most easily prevented. And in preventing injury to toes, often the force of a falling object is so diminished that although the foot may be involved it is not injured. In this connection, there is a word of caution or a discordant note. It seems that many employees have had objects—chiefly rolling slate or coal—strike their toes with no hurt or discomfort. Eventually they become careless and make little effort to get their feet away from such falling objects, with the result that the whole foot is covered and they sustain fractures of the metatarsals. Several safety directors have found that costs of foot injuries have increased materially on this account. They also report bad practice of resting heavy timbers, etc., upon the toe of the shoe, with resultant damage to it.

In most mines, as the use of safety shoes increased the number of toe injuries greatly decreased. Companies keeping detailed records show 50 to 75 per cent reduction. Usually the steel toe withstands far more than specified; there are records of 5,000 to 10,000-lb. loads that damaged the shoe more or less but left the toes untouched or only slightly injured.

It is interesting to note that the War Production Board, in an amended edition of Conservation Order M-217, issued Oct. 18, 1943, required that safety shoes should be manufactured only in accordance with the "American War Standards Specifications for Protective Occupational Footwear, published by the American Standards Association." This is because the high-grade leather in a durable shoe and the high-grade steel in the toe box are critical materials. It also gives due recognition to the importance of safety shoes in the war effort, through the prevention of many lost-time accidents.

Hand and Finger Protection

Most statistical records indicate that hands and fingers—especially fingers—are more often injured than any other part of body. This is especially true in insured bituminous mines in Pennsylvania, as shown on Tables 4 and 5. No doubt a certain amount of protection is afforded at present, or these injuries would be still greater in number. Most injuries to the fingers are slight, but they often become infected, with serious and occasionally fatal results.

Several safety directors gave the following opinions regarding the use of gloves:

> In handling material and tools, it is much safer to use gloves. Many injuries to feet and legs or from falling are due to insecure holding with a bare hand. In low coal, where men must crawl on hands and knees, knee-pads and gloves are indispensable. Gloves are vital to accident prevention but must be matched to job requirements.
>
> Our hand-loading employees as a rule do not like to wear gloves. The frequency of hand injuries due to the lack of hand protection is high, but as a rule these are not too serious. Hand protection should be used.
>
> It is my opinion that the type of loose fitting, poorly formed canvas and other composition glove materials manufactured for the purpose of retailing at a low cost, promotes finger injuries and the extent of injury to a more serious degree than would occur if no gloves were worn; however, I believe that if a comparatively neat fitting glove made from appropriate grades of leather or other serviceable materials were available at a reasonable cost, finger injuries would be materially reduced.

There was very definite opposition to the use of the gauntlet-type glove, especially in connection with any work around machinery or haulage.

There is little to be added to the opinions just quoted. The need for hand and finger protection is all too obvious. If suitable gloves are available, employees should not be without this safeguard.

Protection from Dust

In recent years most of the mining states have adopted laws that provide compensation for silicosis. To guard against the incidence of silicosis, wet drilling is generally used. Where this is not done, it has become imperative that dust masks be used. It is also highly desirable that dust masks be used by haulage employees on main hauls using considerable sand on the rails as well as by employees handling sand at driers. Workers in very dusty occupations, such as cutting, or at loading points inside or at tipples, are more and more inclined to use dust masks. The filtering out of enormous quantities of coal dust from the air does contribute to healthy lung condition and lowers the incidence of lung diseases among mine employees.

Self-rescuers

The self-rescue mask may not be strictly an article of safety clothing, yet many safety directors believe that it is an essential item of protection for every underground employee.

The U. S. Bureau of Mines reports that about half of the persons who lose their lives in mine disasters are killed by the afterdamp. A large proportion of these men could reach safety if they were able to pass through the area filled with the poisonous fumes.

Some instances are on record where lives have been saved by self-rescuers after explosions and mine fires. If all men were so equipped—and their self-rescuers were in the working place at all times—the possibility of saving lives in these emergencies would be greatly increased. The possession of a self-rescuer should give the miner a greater sense of security. The Bureau of Mines puts it very well when it says, "the self-rescuer is a life saver for the miner, and self-rescuers are as essential in mines as life preservers on ships."

Other Protective Clothing

Loose, ill-fitting, baggy or torn clothing should not be worn, for it is liable to be caught in moving machinery or by mine cars. In the latest publication of the Pennsylvania Department of Mines, for September 1943, is recorded a fatal accident from this cause, as follows: "He was oiling the fan shaft bearing, wearing a sweater which had frayed sleeves. It is evident that the sleeve caught on the shaft and his body was pulled into the shaft." One safety director sums this up in the following statement:

> All of our employees are required to wear close fitting clothing and all of them tie the bottoms of their trousers either with tape, bicycle clips or rubber bands. We have had two accidents from men tying their trouser legs at the ankle and causing a loose blouse at the knee, which caught on some machinery. Since then, trousers are tied at the knee also. Ofttimes this protection is afforded by the use of leggings.

Men who work in low coal and are on their knees much of the time are subject to pain and swelling of the knee, which may become chronic. This condition is compensable in most states, either under the Workmen's Compensation or Occupational Disease Laws. Soft, durable knee pads are available for this type of work and are widely used.

Specifications for Safety Clothing

The National Bureau of Standards has set up specifications and tests for safety hats, goggles and respirators in its handbook H-24—American Standard Safety Code for the Protection of Heads, Eyes and Respiratory Organs. These codes were first issued in 1921 and the latest revision was in 1938.

In 1943 the American Standards Association established the American War Standards code for safety shoes—Specifications for Protective Occupational Footwear.

Both of these codes are detailed and lengthy. However, the Coal Mine Section has specifications for safety hats, goggles and shoes in its Coal Mine Standards. These were first set up in 1932 and are in use today with little change. They are offered here as brief, workable specifications, sufficient to cover the practical requirements of the coal-mining industry.

Safety Hats or Caps

Definition.—The term "safety hat or cap" shall mean any hat or cap, by whatever name it may be known, that is made of such material and in such a manner so as to afford protection to the head from blows, falling objects, electric shock, etc., and be worn by men at work with a reasonable degree of comfort.

A. Types:

A-1. Hats or caps with a complete brim.

A-2. Hats or caps with one or more visors.

A-3. Any combination of a hat or cap.

B. Materials and Workmanship

B-1. Materials used in the manufacture of a safety hat or cap shall be suitable for the particular type, light in weight, water resisting, acid resisting and durable.

B-2. The materials of the lining and the hammock or cradle shall be of the best quality and strength. The assembly shall be so as to give the maximum protection and comfort to the wearer.

C. Construction

C-1. All parts of a safety hat or cap shall be so assembled that those parts coming in contact with a man's head shall have a dielectric strength to withstand 750 volts for at least one (1) minute.

C-2. The lining of a safety hat or cap shall be so assembled that there shall be a cradle or hammock to support the complete hat or cap on the wearer's head. This cradle or hammock shall be constructed so as to be fully adjustable. When this cradle or hammock is properly adjusted, there shall be at least one inch of space between the top of the wearer's head and the top of the inside of the crown. It shall be affixed to the hat or cap in a strong and substantial manner.

C-3. The safety hat or cap shall be so constructed as to comply with the following test: When mounted on a wood block similar to the position on a man's head, it shall withstand a pressure blow from an 8-pound iron ball (approximately 4 in. in diameter) dropped vertically on to the center of the crown from a height of five (5) feet without denting or breaking sufficiently to touch the wood block. The cradle or hammock shall withstand the impact from this same blow without breaking or forcing hat or cap down over the head.

C-4. All safety hats or caps shall be fire resisting.

C-5. The safety hat or cap shall be so designed and constructed as to give the required protection and be worn with a reasonable degree of comfort.

Safety Goggles

A. Types

A-1. The cup type goggle and/or its modifications.

A-2. The spectacle type goggle and/or its modifications.

B. Materials and Workmanship

B-1. Materials used in the manufacture of goggles shall be suitable for the purpose, of enduring quality, capable of being sterilized without deterioration and nonirritating to the skin when subject to perspiration.

B-2. All metal materials used shall be corrosion resistant.

B-3. Nitrocellulose or materials equally inflammable shall not be used.

B-4. Materials used shall be such as to combine mechanical strength and lightness of weight to a high degree and workmanship shall be the best throughout.

B-5. Where cup type goggles are used, the cups shall be ventilated. The edge of the eyecup shall form a cushion at least

3/16 in. in facial contact width and shall be so designed as to prevent cutting of the face. If metal eyecups are used, the facial contact edges shall be bound with a cushion at least 3/16 in. in facial contact width and shall be so designed as to prevent cutting of the face by contact with the metal cup edge. This binding shall be non-flammable.

B-6. Head bands shall be of good grade, at least 3/8 in. wide. They shall be adjusted to length and shall be fastened to the eyecups or goggle in such a manner as to be easily replaced and shall not interfere with the facial fit of the eyecups.

C. Lenses

C-1. All lenses shall be made from a single solid glass plate of a quality suitable for optical use and shall be capable of passing the test as to quality, visibility and shock resistance of the Department of Standards of the Federal Government.

C-2. The strength of all lenses shall be such as to comply with the drop ball tests as required by the Department of Standards in the Federal Government specifications.

C-3. The optical surfaces of all lenses shall be free from visible surface defects. They shall be ground or polished on both sides. They shall not be effectively out of parallel. All lenses shall be suitable for continuous use by the wearer without discomfort or optical impairment to the normal eye.

C-4. All lenses in every type of goggle or eyecup shall bear some permanent distinctive marking by which the manufacturer may be readily identified.

D. General

D-1. The specifications herein set out in Items *A* to *D* are abridged from the National Bureau of Standards Handbook H-24, American Standard Safety Code for the Protection of Head, Eyes and Respiratory Organs, issued Nov. 1, 1938.

E. Use

All employees, when using hand tools, shall wear safety goggles.

The term "hand tools" shall mean all hand tools except shovels used in and about a mine. This includes hammers, chisels, hatchets, axes, saws, picks, bars, wrenches, jack hammers, air drills, emery wheels, and all kinds of tools, whether the same requires one or two hands in their practical use.

Safety Shoes

Definition.—The term "Safety Shoes" shall mean any shoe, boot or pac that has a protective toe cap inserted in the toe so as to give a reasonable amount of protection to the foot.

A. Types

A-1. Leather shoe, boot or pac.

A-2. Rubber shoe, boot or pac.

A-3. Any combination of leather and rubber shoe, boot or pac.

B. Materials and Workmanship

B-1. Materials used in the manufacture of "safety shoes" shall be suitable for the particular type and of high grade, so as to give the maximum service.

B-2. The protective toe cap shall be constructed of material that will not be affected by heat or moisture conditions.

C. Construction

C-1. The protective toe cap shall be so inserted in the shoe that it becomes an integral part of the shoe and shall not be a floating cap, or what may be styled an appendix to the original shoe.

C-2. The protective toe cap shall be arched or shaped and inserted so as to withstand 3000 lb. compression and 50-ft.-lb. impact, as set out in American War Standards test of safety shoes.

C-3. The protective toe cap shall be large enough to cover at least three toes.

Note: Some protective toe caps are arc shaped and long enough to cover all five toes.

CONCLUSIONS

Best results have been achieved in the use of safety hats, probably because often serious and fatal injuries have been prevented by such hats and therefore the men are anxious for this protection.

The use of safety shoes has been increasing in a satisfactory manner. However, it should be the aim of the coal-mining industry to have every employee wear safety hat and shoes.

The experience with goggles has been disappointing. There is a wider distribution of goggles but no greater use of them than for some years previous. Until employees are more concerned as to the utter loss entailed when sight is gone, they will continue to risk and to have many needless eye injuries.

Dust masks and gloves are being used where necessary by many men. Care is also being given to the general wearing apparel. It is to be hoped that the men themselves, in the not too distant future, will seek and demand every possible protection afforded by safety clothing.

DISCUSSION

(*Richard Maize presiding*)

J. T. Ryan, Jr.,* Pittsburgh, Pa.—I think Mr. Berner has prepared an excellent paper in a difficult field. Statistics of all kinds are available these days, but it appears that accurate and reliable statistics in the field of safety are often very difficult to obtain, particularly in such a specialized subject as just what accidents have been prevented by the use of safety equipment.

I think also that Mr. Berner has been unduly modest in not emphasizing the work of the Pennsylvania Compensation Rating Bureau in the scheduled credits they have established for the use of safety equipment. This is a very excellent practice and one that has contributed greatly to the progress of safety in the smaller and medium sized mines in the state. It has been a very worth-while step.

In reviewing Mr. Berner's paper, I was impressed by the number of companies reporting that they had their men 100 per cent equipped with safety shoes, protective caps and goggles. Yet it was commented upon in the paper, and it is obvious from these figures, that the use of goggles has not led to the same proportionate reduction in eye injuries as have protective caps and safety shoes in head injuries and toe injuries.

I doubt if the actual use of goggles is nearly as high a percentage as shown by these figures. In other words, goggles may be furnished to all the men in a mine but at the point of an accident perhaps only some of the men are actually wearing goggles; whereas it is reasonable to assume that almost all the men that have shoes and protective hats use them. There may be a sound basis for this. Most of us wear hats and shoes, and it is more likely that the men through custom will wear protective hats and safety shoes, particularly since their weight has been reduced and their wearing comfort has been increased. In any event, they are much more difficult to set aside than are goggles. You can put goggles in your pocket or push them up on your forehead, but it is a little more difficult to dispose of a protective hat or a pair of shoes.

Mr. Berner has mentioned this problem of the use of goggles, and it is a serious one. I believe it leads to a broader problem today—the problem of vision generally underground. A great many people are taking decided interest in that problem today. The increasing use of mechanical equipment underground has brought many mining operations more closely to approximating factory or manufacturing conditions in general, and the problem of vision is receiving greater attention. Although better lighting through the use of improved electric cap lamps is a contributing factor to better vision, many companies provide for eye examinations and, where necessary, underground employees are fitted with prescription goggles with hardened protective lenses. The Union Pacific Coal Co. some years ago undertook a broad program along these lines, and I believe that could be followed with success by a great many other mining companies.

Face shields were mentioned. These are plastic and as yet there have been no definite standards in this field. Committees are working on that. But there has been such an advance in technology in the plastics industry that I believe we can hope for continued progress in the application of transparent protective plastics to the problem of eye protection underground. This may in some measure provide the answer to the problem of eye injuries, which is a serious one.

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* Mine Safety Appliances Company.

J. J. RUTLEDGE,* Annapolis, Md.—Mr. Berner's paper contains a great deal of factual tabulation that should be made available to coal mining officials and state mine inspectors. He has had access to valuable sources of reliable information, and is to be commended for collecting and tabulating the data.

When a coal company has provided, or is ready to provide, at cost or less than cost, goggles, hard hats, hard-toed shoes, the employees should do their part in the adoption of these safety measures. However, it is sometimes difficult, or actually impossible, to secure the active cooperation of the employees, and sometimes of the operators, in the introduction of these devices.

For example, in our state there are two machine mines operating in the same seam and adjoining each other. In one of these mines the miners wear hard hats, electric cap lamps and hard-toed shoes, and, to some extent, goggles. In an adjoining mine—in the same seam, be it remembered—the miners and mine workers absolutely refuse to adopt these safety devices. The success of the effort in the first mine is largely due to the diplomacy and safety mindedness of the foreman.

In another mine, which is just about five years old, the young foreman and superintendent, both of whom are products of the Maryland night mining classes, decided when the mine was opened that no employee should enter unless he was equipped with electric cap lamp, hard hat and hard-toed shoes, and this mine is 100 per cent equipped. Incidentally, it is one of the most efficiently operated mines in the state.

Another large mine, belonging to one of the largest companies, has had electric cap lamps on hand for a long time but has not been able to secure their adoption. Both of these mines operate in relatively thin coal seams.

The plastic eye guard is one of the best safety devices that has been introduced in mines in recent years. Not long ago the writer, while passing a marble-cutter's yard, noticed the proprietor cutting a piece of sandstone, rounding it into shape to serve as a cornerstone. This was a hurry-up job and the work was being done in a yard fronting on the principal street of the mining town. The stonecutter was not wearing goggles. The writer, knowing him quite well, spoke to him about the danger he was running in cutting stone without goggles and the risk he was taking, but the stonecutter stated that this was a hurry-up job and the government wanted it in a hurry. About two weeks afterward the writer passed the same yard, saw the same stonecutter at work and spoke to him, and noticed that the cornea in his left eye was wilted, and on inquiry learned that the stonecutter had lost the sight of the eye, a chip having flown into it while he was at work.

Another instance is that of a young woman, a seamstress working in a clothing store, who was severely burned by an explosion of natural gas. At the time of the accident she was wearing ordinary spectacles. Her body was nearly covered with first, second and third degree burns. She was hospitalized but died about four or five days afterward. The surgeon in charge assured the writer that had the young woman survived the burns, she would have had full use of her eyes because she was wearing glasses. He remarked that during all the time he was treating the burns her eyes never lost sight of his hands.

Two young men in a fire-clay mine were mucking out fire clay blasted by a previous shift, and started their jackhammers before all the loose fire clay had been shoveled out. They were being rushed for clay. The drill struck an unexploded charge and both the men suffered damage from the blast. One of them had eight punctures through the right eyeball. The helper was badly shocked but his eyes were not seriously damaged. These men were taken immediately, a distance of 150 miles, to a hospital and were given the best surgical treatment that was available in the city. The writer visited them in the hospital. The eyes of both men were covered and the one with the punctured eyeballs said, "Oh! we made a mistake." This man lost the sight of both eyes. He can detect the difference between night and day but that is all. Had he been wearing a cap equipped with a plastic eye guard, or with ordinary goggles, in all probability he would not have lost his sight.

E. MCAULIFFE,* Omaha, Nebr.—I have not had an opportunity to read Mr. Berner's paper in full. Glancing at it, I have no hesitancy in saying that it is a masterful review of the safety situation.

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* Chief Mine Engineer, Maryland Bureau of Mines.

* President, The Union Pacific Coal Co.

I wish however to bring you a word of encouragement. There must be something wrong with the attitude of mind of the average operator toward this safety question; otherwise, more would be accomplished. I had considerable concern about what was to happen on our properties in the last couple of years where we have had a labor turnover of 115 per cent. Perhaps 30 per cent of the men that we are forced to employ—and they all came certified by the War Man Power Commission—were either partially moronic or psychopathic and definitely difficult to deal with. Nevertheless, we have been able to put safety clothing on every last one of them. Some of them stayed with us only two or three days and some men carried the clothing away with them. Others pawned it and went out of town on what they got for the second-hand clothing. But we have never deviated from our standard for one moment and that includes not only goggles, but goggles ground for sight correction. Many of these men did not stay until we could get the corrected goggles out to them, but the point I am making is that we have held to the principle.

There is no man working in our mines that does not wear goggles. A man cannot work there without wearing the goggles. In the beginning we met with all sorts of resistance. We tried to handle the problem judiciously. The first thing we did was to put goggles on our foremen. Perhaps they thought they had to work for the company, as they had no union behind them, and they wore the goggles. When men who had the protection of the union —and sometimes it is good and sometimes it is bad—saw these men going along in the even tenor of their way wearing goggles, they came to the conclusion the goggles could not be so terribly bad and they started wearing them, and we had goggles on our property for five or six months before the order making the wearing of goggles mandatory was put out. On the morning the order was given, I can recall very well that A. W. Dickinson, now of the Mining Congress, who was the general superintendent on the property, was morally certain we were going to have a strike. But nothing happened. Every man wore his goggles and the order went through.

As I have said, we have had a desperate time in the last two years trying to maintain a labor force. Still, with 115 per cent turnover last year we have succeeded in keeping safety clothing on everybody and we produced an accident record of 91,600 man-hours of exposure to each lost-time accident. To compare that situation with the first 10 years I was on the property—and I am now in my twenty-second year—we then could not get above about 15,600 man-hours per accident. We are doing almost six times as well now. In the last couple of years we have dropped back from 118,000, and in one instance 124,000 man-hours, to 91,600, but we are still doing pretty well. And I now serve notice on Mr. Dan Harrington that we are out for the Sentinel of Safety Trophy this year, and we hope to get it. That is not, in my opinion, a fair basis of accident comparison because it is applicable only to one certain mine. However I do not object to the method—the point is that I would rather prevent a lot of accidents in nine or ten mines than in one mine.

I also think that the responsibility for safety rests not on the Bureau of Mines, not on the manufacturers that have done such commendable work as Mr. Ryan's company, originated by his father, whom most of us older men knew so well, but with the management of the property.

Our change in man-hours per accident was really accomplished by working on the peculiar psychology of the miner. We started a lottery scheme, and it was really astonishing how the men responded to the chance to obtain an automobile or a cash prize by avoiding an accident when they had been indifferent to injuries to their bodies.

I think that if a little more thought were given by management to the prevention of accidents, the opposition on the part of workers and the union, which is very real, could be overcome and betterment brought about.

C. E. Berner (author's reply).—Mr. McAuliffe has very definitely stated that this is a management proposition, and I think it has been for many years. I have suggested in the paper that it has become of recent years increasingly a problem of labor. The men are more assertive than heretofore. Some of you know better than I the steps that have been taken to incorporate active participation on the part of labor organizations in the safety program. I do not just know who is the most active sponsor. I know Mr. Thomas Moses has given a good deal of thought to that; and through the National Safety Council, the labor unions have not only been invited or urged but have accepted membership on various safety committees. That will take some time to filter down through the ranks and all of that work I presume has been more or less delayed by the very chaotic wage negotiations and situations in the last six months or more. I am not at all prepared to say that management has gone nearly as far as it can or should, but great help can be achieved through an educational program on the part of responsible labor representatives. It seems that the average working man is far more ready to listen to his committee man or some organizer than he is to his foreman or superintendent. I want to re-emphasize that as being a real hope for the future.

The Recovery of Pyrite from Coal-mine Refuse

By David R. Mitchell,* Member A.I.M.E.

(New York Meeting, February 1944)

The mineral pyrite (or marcasite) occurs in coal beds as balls, lenses, veinlets and bands. Several million tons are wasted annually on the refuse dumps from coal mining and coal-preparation activities. Sporadic attempts have been made in the United States to recover this pyrite for manufacture of sulphuric acid but only a few plants have succeeded in recovering it on a commercial scale.

Several chemical companies in the East became concerned about supplies of pyrite early in 1943, but by late summer of that year the situation had eased.

If for no other reason than national security and self-sufficiency during wartime, methods of recovering pyrite from coal-mine refuse should be worked out and made available to coal-mine operators who are potential producers of pyrite. Eastern sulphuric-acid plants are particularly likely to have their sources of raw material blocked by war conditions, so that special studies to determine where supplies of pyrite can be obtained for these plants from inland sources are desirable.

Davis's[1] summarization of conditions during World War I gives the following facts: During those years, the shortage of pyrite became acute. Shipments of hand-picked pyrite were made from a number of coal mines in Illinois, Indiana, western Kentucky, Ohio and Pennsylvania. A mechanical concentrating plant for separating pyrite from picking-table refuse was built at Danville, Ill., and operated for several years, and a small pyrite-concentrating plant treating washery refuse near Gillespie, Ill., operated for several months in 1918. The Consolidation Coal Co. of Fairmont, W. Va., constructed and operated a sulphuric-acid plant using pyrite recovered from coal-mine refuse. The U. S. Bureau of Mines during 1917–1919 made a comprehensive survey of pyrite-production possibilities from coal mines and estimated that there was a potential yearly production of 435,000 tons from Ohio and Pennsylvania and 928,000 tons from Kansas, Missouri, Iowa, Illinois, and Indiana mines. This estimate included only mines that could produce one per cent or more of their coal production as pyrite containing a minimum of 40 per cent sulphur.

Pyrite Production

According to the Minerals Yearbook,[2] approximately 660,000 long tons of pyrite was produced in the United States in 1941, with an average sulphur content of 42 per cent. The average value f.o.b. mines was $3.09 per long ton. Spanish pyrite, 48 per cent sulphur minimum, was quoted at 12 cents a unit c.i.f. U. S. ports, equal to $5.76 a ton for shipments of minimum sulphur content.

Most of the pyrite produced in the United States is obtained as a by-product in the dressing of copper and zinc ores. Metal mines in the East making pyrite concentrate for the market or use in company-operated sulphuric-acid plants

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[1] References are at the end of the paper.

consist of two zinc mines in St. Lawrence County, New York State, a magnetite iron-ore mine in eastern Pennsylvania, and a pyrite mine in Virginia.

Shipments of pyrite from coal mines during 1941 were reported as follows:

Illinois.—The Midland Electric Coal Corporation, Henry County, shipped 12,026 long tons containing 46 per cent sulphur, at its Atkinson preparation plant.

Indiana.—The Snow Hill Coal Corporation produced pyrite from its Talleydale preparation plant in Vigo County.

Kansas.—The Mineral Products Co. produced 3902 long tons of pyrite containing 44 per cent of sulphur from picking table and washery refuse.

SPECIFICATIONS

Specifications for coal pyrite for acid manufacture range from 5 per cent to 8 per cent maximum carbon limit, and 42 to 45 per cent minimum sulphur limit. Where flash roasting is practiced, the pyrite must be minus 100-mesh. Most companies give no size specifications. One company specifies 20-mesh or finer. Impurities in pyrite other than carbon may limit its use—for instance, arsenic, probably as arseno pyrite—even though a concentrate can be made complying with specifications otherwise.

CHEMICAL CONSTITUENTS OF PYRITE FROM COAL BEDS

Although much has been written during the past 25 years concerning pyrite occurring with coal, few published analyses are available. Table 1 gives an indication of the various constituents likely to be found in lump pyrite and in concentrates.

PYRITE-RECOVERY PLANTS

Bradford breakers and picking tables may be used to make a marketable product where the pyrite occurs largely in sizes coarser than about 1 in. To make a grade of concentrate acceptable to sulphuric-acid plants the pyrite must break freely from the accompanying coal and slate, which is a rare condition to find. The flow of material in the plant shown in Fig. 1, described by Verhoeff,[4] is typical of plants of this kind.

TABLE 1.—*Analyses of Pyrite*
PER CENT

Constituent	1[a]	2	3
Total sulphur	45.28	43.36	50.54
Total carbon	4.12	8.83	
Moisture	3.80		
Sulphate sulphur	0.23		
Iron	40.90	37.44	41.85
Lime	2.40	0.68	
Insoluble	1.54		
Copper	0.52		
Arsenic	0.004	0.12	
Zinc	0.19		
Lead	0.17		
Silica	0.27	2.51	0.385
Undetermined	0.576		
Alumina		4.97	
Magnesium oxide		0.51	
Phosphorus		0.05	
Hydrogen		0.73	
Bituminous matter			7.52

[a] 1. Concentrate from the Mineral Products Co. mill at West Mineral, Kansas.[5]

2. Concentrate from a laboratory test of refuse obtained at a western Pennsylvania coal-preparation plant cleaning Pittsburgh-bed coal.

3. Lump pyrite reported by Stephen P. Burke and Richard Downs.[3]

Most pyrite occurring in coal beds requires crushing and an abrasive type of grinding to free it of entwined coal and slate, followed by mechanical concentration by jigs, tables or other concentrating devices suitable for heavy ores, to make a satisfactory concentrate. At the Snow Hill Coal Corporation's plant in Indiana and at the Midland Electric Corporation's plant at Atkinson, Ill., the final concentrate is made by a mechanical jig.

The most recent and complete mill for concentrating pyrite from coal-mine refuse is that of the Mineral Products Co. at West Mineral, Kans. Details of this plant have been reported by K. A. Spencer.[5] The flowsheet is shown in Fig. 2 and the over-all operating economics are given in Table 2. Many regrinds and recirculations of products are necessary to get a concentrate of uniform grade.

The general principles shown in this plant are sound. It is too complicated for general adoption in the coal-mining industry, but developments in the art of mineral preparation have been rapid, and it is likely that a much simpler plant could now be designed, incorporating the general principles shown in this plant to be necessary to get a satisfactory grade of concentrate.

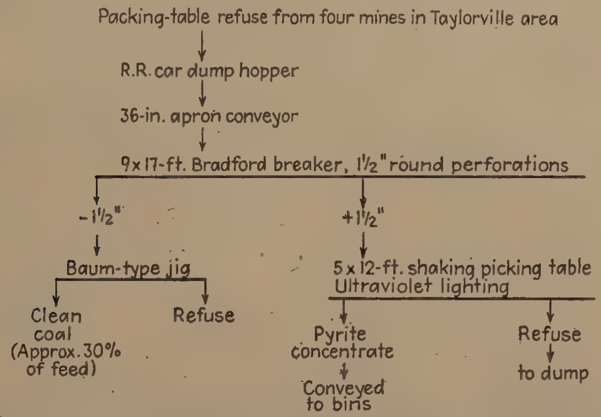
Packing-table refuse from four mines in Taylorville area
R.R. car dump hopper
36-in. apron conveyor
9 x 17-ft. Bradford breaker, 1½" round perforations
-1½"
+1½"
Baum-type jig
5 x 12-ft. shaking picking table
Ultraviolet lighting
Clean coal (Approx. 30% of feed)
Refuse
Pyrite concentrate
Conveyed to bins
Refuse
to dump

FIG. 1.—SIMPLIFIED FLOW DIAGRAM, PEABODY COAL CO., TAYLORVILLE, ILLINOIS.

It is apparent that the feed to this plant could not have contained more than 5 per cent sulphur, or approximately 10 per cent of pyrite.

TABLE 2.—*General Operating Economics, Mineral Products Company, West Mineral, Kansas*[5]

Operated continuously since July 1936

Plant cost	$170,000
Connected load	312 hp.
Water requirements	3500 gal. per min.
Typical month's operation:	
Total tons milled	9,643
Shifts operated (8 hr.)	29
Hours operated	223
Possible hours	232
Running time, per cent	96.1
Tons milled per hour	43.24
Tons coal produced	2,158
(Equals 22.39 per cent of feed)	
Tons pyrite produced	1,712
(Equals 17.75 per cent of feed)	
Average ash, per cent of coal produced	10.09
Average sulphur, per cent of pyrite produced	45.51
Average realization f.o.b. mill per ton coal	$1.38
Average realization f.o.b. mill per ton pyrite	$2.38

TABLE 3.—*Float-and-sink Tests of Illinois Coal*

Coal Bed No.	Number of Samples Tested	Number of Samples in Which the 1.60 Sp. Gr. Sink Contained More than 5 Per Cent S	Minimum Sulphur Reported in Sink, Per Cent	Maximum Sulphur Reported in Sink, Per Cent
1 and 2	10	10	7.2	34.6
5	25	24	4.9	33.1
6	39	27	0.6	26.2
7	9	9	6.5	25.4
Totals	83	70		

There are many mines at which the refuse produced contains more than 5 per cent sulphur. Mitchell[6] and Mitchell and Smith[7] gave detailed tests on samples of Illinois coal. Twenty samples of sink at 2.97 sp. gr. gave an average sulphur content of 40 per cent. Of 83 samples of Illinois coal tested by float-and-sink methods, the 1.60 sp. gr. sink of 70 of these contained more than 5 per cent sulphur. It would appear that no great difficulty would be encountered in dressing refuse from these coals to a 45 per cent sulphur concentrate. A summarization of these data is given in Table 3.

PYRITE POSSIBILITIES AT EASTERN COAL MINES

In general, coal mines in the Pittsburgh area and to the east of Pittsburgh are not as promising as sources of pyrite as coal mines in the Middle West. Northern West Virginia and occasional mines throughout the bituminous and anthracite regions of Pennsylvania have possibilities. Other sections of West Virginia show occasional high-pyrite areas. There are few high-sulphur areas in Virginia and Maryland.

That there may be a possibility of recovering considerable pyrite from the refuse of small-coal cleaners in the Pennsylvania anthracite region is shown by a test of the refuse from a unit preparing No. 5 buckwheat:

Total weight of sample, lb.	49.6
Per cent on 3/32-in. rd.	3.2
Through 3/32 on 1/16 in. rd.	6.8
1/16 on 3/64 in. rd.	20.2
3/64 on 1/32 in. rd.	44.0
1/32	25.8
Sulphur content, per cent	36.4

If the sulphur content of this sample is all derived from pyrite, which probably it is, the calculated pyrite content of the sample is 68.5 per cent.

There is little large-size pyrite in the anthracite coal beds. Considerable pyrite shows up in some of the preparation plants in the refuse of the small sizes and from a visual inspection appears to be comparatively free of impurities. At plants where there is an appreciable amount of pyrite, it would be comparatively easy to insert a concentration machine in the flow to make a pyrite concentrate, provided a satisfactory market existed.

Tests on a car of refuse from a washery in western Pennsylvania, where minus 4-in. Pittsburgh-bed coal is cleaned, show an appreciable sulphur content in the refuse. The sulphur content increases as the size of refuse decreases (Table 4).

TABLE 4.—TEST OF REFUSE AT A WASHERY IN WESTERN PENNSYLVANIA

Refuse		Ash, Per Cent	Sulphur, Per Cent
Size, In.	Per Cent		
4 to 2½	3.8	64.8	1.44
2½ to 1	26.5	59.8	5.58
1 to 3/8	37.9	56.7	12.96
3/8 to 0	31.8	56.7	13.60

Approximately 70 per cent of this refuse is minus 1-in. and amounts to approximately 3 per cent of the feed coal. On the basis of cleaning 6000 tons of feed coal per day, approximately 45 tons of pyrite is indicated as recoverable. This does not appear to be a large amount, yet if many coal-preparation plants in the Pittsburgh area have a similar refuse an appreciable amount of pyrite could be recovered.

TRENDS

On the whole, the economics of recovering pyrite from coal is not favorable. Certain trends may improve the economics of recovering and using this pyrite but others tend to have the opposite effect.

Favorable trends are the likelihood of an increase in the cost of producing elemental sulphur in Louisiana and Texas. Easily won reserves of Gulf sulphur are being used at a fast rate.

There has been a tremendous increase in the mechanical preparation of bituminous coal during the past 40 years. According to data presented in *Mechannual*,[8] 18,000,000 tons of coal was mechanically cleaned in 1920. By 1941, this had risen to 114,000,000 tons. It requires much less capital expenditure and lower operating costs to insert machines in the flow of an existing coal-preparation plant to recover pyrite than to build a new plant for this purpose.

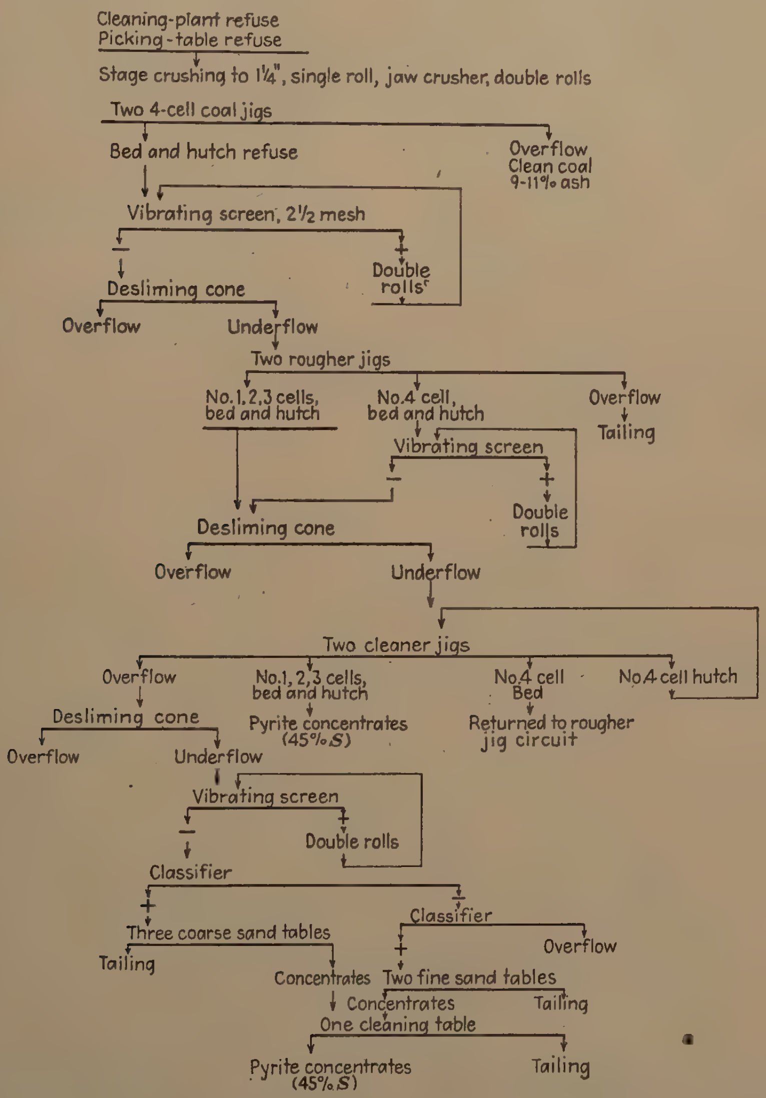

FIG. 2.—SIMPLIFIED FLOW DIAGRAM, MINERAL PRODUCTS CO., WEST MINERAL, KANSAS.

The art of mineral dressing has developed greatly in recent years, and it is technically now much easier to make a pyrite concentrate of uniform grade than formerly. This is both favorable and unfavorable, since more pyrite is being recovered as a by-product from the dressing of metallic ores.

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Separation and Transportation of Fine-mesh Coal Dust as Pulverized Fuel

By H. C. Ray,* Member A.I.M.E.

(St. Louis Meeting, September 1942)

About the beginning of 1930, coal preparation had reached the stage where the cleaning of coal by liquid or air had become the order of the day. Since that time many new preparation plants have been installed in spite of the years of depression in the coal industry, and this is indicative of a trend wherein the coal producer found it expedient to manufacture a cleaner product in order to compete with other fuels.

It was also about 1930 that, because of the difficulties encountered with fine coal in cleaning plants, dust collectors came into use. Both cleaning plants and dust collectors, therefore, were much discussed at the engineering meetings of the early 1930's. During these meetings the many advantages to be gained through the use of dust-collecting systems were pointed out by Hebley,[1] Appleyard[2] and others. The great possibilities of new markets for this product were emphasized and discussed at the same time; however, the disposal of the collected dust remained a problem.

Since the purpose of dust collecting is to improve the preparation and, consequently, the salability of the coal, it seems illogical to return the collected dust to any of the coal being prepared for market use, as is done in preparation plants that include dust-collecting systems.

The logical thing is to dispose of the dust in some useful and practical manner outside of the tipple. As a practical matter, this can be done in one or both of two ways; either by consuming the dust at the mine's own power plant or by marketing the dust direct to the consumer. The latter method not only offers the greatest opportunities, but is the most important, because of inequalities that may exist between the quantity of dust collected and the consuming capacity of the mine's power plant.

In considering the more important marketing possibilities, we are confronted immediately with the problems connected with the handling and transportation of so fine a material. It is this problem, rather than the problem of finding new markets, that has been responsible for the lack of progress in the matter of dust disposal.

It is the purpose of this paper to point out again some of the potential market outlets, and also to describe an existing system of handling and transporting coal dust. To the best of the writer's knowledge, this is the first and only operation in this country that has handled fine-mesh coal in bulk 50-ton lots direct to the consumer.

POTENTIAL MARKETS FOR FINE-MESH COAL DUST

In general, the potential market outlets would include any operation of which the pulverization of coal is a necessary part. To be more specific, a few of the better known operations of this type must include: colloidal-fuel plants, briquetting plants, "sea coal" distributing plants, water-purification plants and, obviously, pow-

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[1] References are at the end of the paper.

dered-fuel power plants. Powdered-fuel plants, because of their greater number, their type of equipment and their greater consuming capacity, probably offer the most practical outlet, and certainly they furnish the greatest immediate market. It should be pointed out, however, that many of the other outlets that are not as yet fully developed would progress much faster if fine coal dust were made available to them.

The development of the burner to use powdered coal for small commercial and domestic heating is one outlet, for example, that has very definitely been held back because of unavailable fuel. In trying to compete with other fuels, a number of small unit pulverizers and burners have been developed, but have been found impracticable because of noise, adjustments and the dangers involved when they are placed in the hands of the average consumer. Practical burners of the storage-bin type have also been developed but are unmarketable because of the lack of pulverized fuel. If powdered coal were available to local dealers in carload lots, these burners would be entirely practical, in the writer's opinion, because coal dust can be handled in equipment similar to that now used in distribution of fuel oil. Coal dust can be handled from tank truck, through hose or pipe, to the consumers' bins with all the convenience and cleanliness of oil fuel.

DEDUSTING OF COAL

During the past several years the Carter Coal Co. has been collecting dust at one of its mines in the Pocahontas No. 4 seam, McDowell County, West Virginia. The coal mined there is a low-volatile coal low in moisture and very friable. At that mine, the dust-collecting system was not installed in combination with any washing or cleaning operation, but was meant to improve the prepared sizes and to eliminate hazardous working conditions around the tipple. Briefly, the dust-collecting system to be described accomplishes the following results:

1. It removes float dust from the tipple, thereby reducing maintenance cost and hazardous working conditions.
2. It removes excess dust that normally would stay with the prepared sizes in spite of the screening operations.
3. It recovers tons of dust normally lost to the ground and surrounding countryside.
4. It prevents dust from settling on prepared sizes after they have been loaded into railway cars.
5. It prevents dust from settling in empty cars waiting to be loaded.
6. It receives dust from the vibrating screens, which otherwise would find its way into the small resultant.
7. It removes fine float dust from the fine slack left from the screening of small stoker-coal sizes.
8. Finally, it produces a product consisting of 100-mesh to zero dust, ready for market.

Dust-collecting System

The selection of dust-collecting equipment and the methods used to extract the dust from the coal naturally depend upon the specific needs of the individual mine. Many of these standard methods are ably described by H. F. Hebley.[3] The system to be referred to in this paper, however, consists of a suction pickup system, cyclone collectors, cloth collectors, a pulverizing mill, and a Fuller Kenyon air-pump system for handling the finished product through storage bins, which will be referred to as the "fine-dust bin," the "final storage bin" and the "coarse-dust bin." The dust is first picked up by suction from completely housed terminals in the tipple at strategic points in the coal flow. From these terminals the dust is conveyed through pipes to the cyclone collector, where the coarse dust is precipitated and passed down to the coarse-dust bin. The exhaust from the cyclone collector passes through a pipe to the baghouse, where all the dust is removed from the air by means of cloth screens; then the clean air is exhausted to the atmosphere. The dust from the baghouse is 200-mesh and finer, needs no further preparation, and is taken by screw conveyor to the fine-dust bin.

The baghouse dust can be pumped from the fine-dust bin to the mine powerhouse, or over to the final storage bin. The dust from the coarse-dust bin can be pumped into the final storage bin, by gravity to the railroad car, or by gravity taken down to a ball-mill pulverizer, where it is classified and ground to 100-mesh to 0. After pulverization it can be pumped either to the mine powerhouse or on to the final storage bin. As a safety measure, the equipment is designed so that the dust in any of the bins can be circulated by taking it by gravity from the bottom of the bin and pumping it in at the top. Should heating occur in any bin, this circulation system provides a quick means of cooling. If the tipple is not in operation, slack coal to be pulverized can be unloaded to a railway track hopper, fed to an elevator, discharged to the coarse-dust bin and thence to the pulverizer.

The flexibility of this system permits the collection and shipment of dust in its raw state or pulverized to any reasonable degree of fineness needed to satisfy a customer's demands. Its bin capacity is sufficient to take care of normal demand, even with intermittent operation of the mine. Its design permits slack coal to be pulverized when dust is not being collected from the tipple.

Finally, for shipment to market the cars are of the type used for shipping cement, and the dust is simply loaded by gravity through flexible canvas tubes connected to the hatch openings in the top of the car.

Transportation

As mentioned, probably the greatest restraining factor in the use and distribution of powdered coal dust prepared at the mine is the problem of transportation. The smallest sizings of pulverized bituminous coal, in this case 100-mesh to zero, obviously cannot be transported in ordinary railroad cars. When loaded, pulverized bituminous coal is well aerated, whether it is loaded by gravity or by means of compressed air. The result is that the material flows into the car and, like water, seeks its own level. This means that there is no difficulty in completely filling all spaces, regardless of the shape of the container. The vibration of the railroad car in transit causes the dust to pack down to a consistent mass, which will support the weight of a man and barely shows his footprints on the surface. Hence the ideal car for transportation of this material is one that can hold enough aerated coal dust to meet the weight requirements of the railroads, so as to avoid freight charges in excess of those normally charged per ton of coal. Furthermore, the cars must be so constructed that they will prevent any pickup of moisture, loss through windage or loss through leakage. Also because of the nature of the material, the cars must be easy to unload, therefore should contain convenient outlets supported by a sloping bottom, equal to or greater than the angle of repose of the dust, which in some cases reaches as much as 60 per cent.

In Europe, particularly in England and Germany, the transportation of coal dust is an old story, and engineers there have developed two types of cars—a tank car and a car that carries two or three separate vertical round tanks or tubs.[4] However, European conditions and railroad facilities are quite different from those in America, and in America neither of these two types is suitable.

The standard tank car may be filled easily enough but the average cubical content is not great enough to hold a satisfactory load even if vibration were applied during the loading process. Furthermore, there are not enough outlets in the standard horizontal tank car, and even if they were provided it would be impossible to completely empty the car.

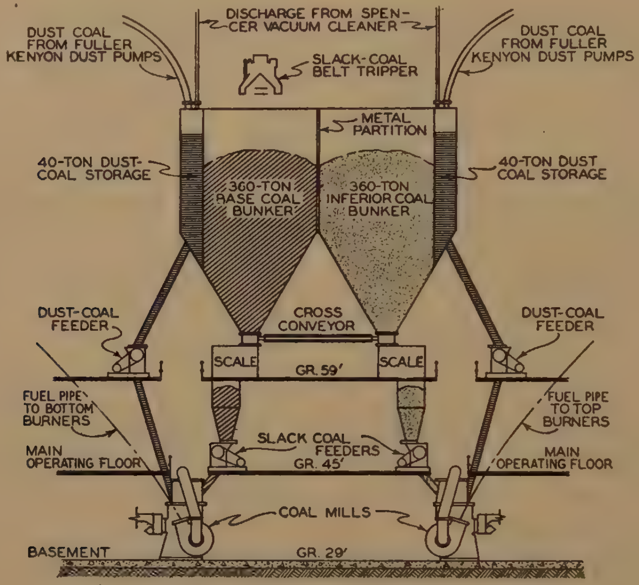

Fig. 1.—Slack and dust-coal bunker system.

The tub cars are not standard equipment in this country, therefore are not available, and they too, owing to their construction, are limited in capacity. However, if the distribution and marketing of fine coal dust increases in amount, the tub cars may play their part here, because the individual tanks or tubs can be removed and would serve as temporary storage bins for small consumers.

Earlier than 1930 the demand for bulk shipments of cement and other pulverized materials in this country caused the railroads to develop a car expressly for the purpose, commonly known as the cement car. During the depression years, the construction of these cars was retarded and most railroads do not have enough of them to satisfy the demand. However, the trend in this direction makes it easier to solve the difficulty of coal-dust transportation because these cars are very satisfactory for the purpose. Since they were primarily designed for the transportation of cement, their capacity is generally about 50 tons, and this quantity probably would necessitate the use of vibrators when loading coal dust, in order to make a satisfactory railroad pay load. Because of the increased demand for this type of car, at least one railroad has built cars having 70 tons capacity, which enables them to carry satisfactory loads of the lighter density materials such as pulverized coal dust. These cars are owned by the railroad that services the only mine from which bulk pulverized coal has been shipped successfully, and the writer's experience enables him to say definitely that their performance has been entirely satisfactory.

Unloading and Consumption

In seeking a possible market for the 100 mesh to zero coal dust, the writer visited a number of pulverized-fuel plants conveniently located along the originating railroad. It was an agreeable surprise to note the great interest shown in virtually every plant, particularly those having storage-bin systems. This is understandable, since one of the advantages of mine-prepared pulverized fuel is the elimination of the substantial cost connected with the drying and pulverization of slack coal necessary with the storage-bin system. Strange as it may seem, a contract was made for the use of the dust with a large utility company having a new and most modern unit pulverizer system. It stands to reason that if a plant of this type finds it advantageous to use mine-prepared coal dust, great market possibilities are indicated even if only the one field of pulverized-fuel plants is considered.

After preliminary studies were made by the power plant mentioned, which indicated that the use of 100 mesh to zero coal would be advantageous to them, the necessary equipment for unloading, conveying and burning the coal dust was installed. This first experimental equipment consisted of a track unloading pit containing a Fuller Kenyon pump, a 40-ton storage silo in the slack-coal bunker, and a table feeder feeding to a unit mill pulverizer. While the dust actually needs no further pulverizing, it was considered good practice to feed the coal through the unit mill, thereby eliminating the expense of further changes in the existing equipment, and also to ensure the pulverization of any coarse coal that might find its way accidentally into the original dust. The hopper bottoms on the cement-type car are equipped with sliding gates. To overcome the difficulty of getting the dust to flow after it has settled into a solid mass during transit, a Syntron vibrator is attached to the side of the car and remains in operation until the car is unloaded. The gate outlets are arranged so that a canvas boot can be connected between the gates and the Fuller Kenyon unloading pump. The contents of the car are conveyed from the pump through pipes to the overhead storage silo. From the silo the dust can be used alone to operate the boilers, or it can be used to supplement a pulverizer while it is in operation with slack coal. This installation (Fig. 1) was made in 1939, and no operating difficulty has been experienced in its use.

During the course of operation, an analysis was made of the coal dust as received (Table 1).

TABLE 1.—*Analysis of Coal Dust as Received*

Proximate Analysis		Screening Analysis, Per Cent	
Moisture, per cent	0.37	Through	
Volatile matter, per cent	16.35	200-mesh	93
Ash, per cent	6.70	100-mesh	97
Fixed carbon, per cent	76.58	60-mesh	99.6
Heat units, B.t.u.	14,590	40-mesh	99.9
Sulphur, per cent	.52		
Ash-fusion temperature, deg F.	2350		
Ash-fluid temperature, deg F.	2450		

CONCLUSION

With the trend in coal preparation constantly moving toward the use of smaller sizes and the demand by consumers for dustless fuel, it is almost certain that the practice of dust collecting will become a necessary step in the normal preparation of coal. The disposal of the collected dust will become something more than just a nuisance problem, and, using present railroad facilities, this new product will find its way into the great potential markets which for years have been well recognized.

ACKNOWLEDGMENTS

The author acknowledges with thanks the aid given in the preparation of this paper by J. A. Reich, Superintendent of Power Plants, Virginia Electric Power Co., and by R. A. Van Dyke, Mining Engineer, Carter Coal Company.

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Occurrence of Phosphorus in Washington Coal, and Its Removal

By M. R. Geer,* Member, Franklin T. Davis,† Student Associate,
and H. F. Yancey,‡ Member A.I.M.E.

(New York Meeting, February 1943)

Coke with low phosphorus content is required by some of the electrometallurgical and chemical plants recently attracted to the Pacific Northwest by the hydroelectric power available from Bonneville and Grand Coulee dams. Shipping such coke into the area from eastern points involves a high freight charge, and during wartime consumes precious rail and ship-bottom space, so it is highly desirable to produce coke from the coking coals of Washington to meet the requirements of these industries.

Although coke has been made from Washington coals since 1884, little has been known about their phosphorus contents or about the extent to which phosphorus could be reduced by cleaning. An investigation was undertaken, therefore, to determine the manner in which phosphorus occurs in the coking coals of Washington and its amenability to removal by ordinary coal-washing methods.

Previous Work

Phosphorus is a deleterious element in coke used for numerous metallurgical processes, including the production of pig iron, yet comparatively little published information is available on the occurrence and elimination of phosphorus in any of the coals of this country. A variation from 0.009 to 0.062 per cent phosphorus in various horizons of the Pittsburgh bed in the Connellsville region of Pennsylania has been reported,[1] and a reduction in phosphorus content from 0.020 per cent in the raw coal to 0.018 per cent in the washed coal was obtained in washing coal from the same bed in the Pittsburgh area;[2] the latter figures represent average analyses for a year's operation of the washery on about 800,000 tons of coal. A reduction in phosphorus content from 0.018 to 0.011 per cent in washing coal from the Pittsburgh bed at another washery was observed, and the same report provides data on the phosphorus reduction obtained by separating 10 other coals on a bath of 1.35 specific gravity.[3]

European investigators have been especially active in correlating phosphorus content with petrographic analysis; fusain and durain generally contain the highest percentage of phosphorus, but this relationship is not universal.[4] Some of these investigators have found that phosphorus is removed in the coal-washing process,[5] but others have observed that the clean coal contained as much or even more phosphorus than the raw coal.[6]

This report gives the results of work done under a cooperative agreement between the U. S. Department of the Interior, Bureau of Mines, and the College of Mines of the University of Washington. Published by permission of the Director, Bureau of Mines. Manuscript received at the office of the Institute Dec. 1, 1942. Issued as T.P. 1586, July 1943.

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[1] References are at the end of the paper.

PLAN OF WORK

From the standpoint of removal by ordinary coal-washing processes, the specific-gravity distribution of phosphorus is of foremost importance. If the phosphorus is associated with the heavy impurities in a coal it can be eliminated in the washing operation to the same extent that these impurities are eliminated; if, on the other hand, it is distributed throughout the light, low-ash coal it cannot be removed by washing. Therefore specific-gravity analyses were made on representative samples of the most important coking coals of Washington, and the phosphorus in the individual density fractions was determined. Each sample was mined from the coal bed in such a way as to include all of the impurities that would have to be removed in a commercial washing operation, and the coal was tested at about the size to which it would be crushed for washing. Actual washing tests on a full-size wet table augmented the specific-gravity analyses.

TABLE 1.—*Specific-gravity Analysis of Coal from Wilkeson West 3 Bed, Crushed through ¼ Inch*

Size	Specific Gravity	Weight, Per Cent	Ash,[a] Per Cent	Phosphorus,[a] Per Cent	Cumulative, Per Cent		
					Weight	Ash[a]	Phosphorus[a]
¼-in. to 28-mesh. Weight, 84.0 per cent.	Under 1.40	50.0	9.2	0.044	50.0	9.2	0.044
	1.40–1.50	10.7	24.5	0.040	60.7	11.9	0.043
	1.50–1.60	6.3	34.7	0.061	67.0	14.0	0.045
	1.60–1.70	4.0	43.4	0.084	71.0	15.7	0.047
	Over 1.70	29.0	71.3	0.360	100.0	31.8	0.137
Under 28-mesh. Weight, 16.0 per cent.	Under 1.40	40.6	8.7	0.041	40.6	8.7	0.041
	1.40–1.50	12.6	20.6	0.055	53.2	11.5	0.045
	1.50–1.60	8.0	34.3	0.082	61.2	14.5	0.051
	1.60–1.70	6.7	40.5	0.094	67.9	17.1	0.054
	Over 1.70	32.1	67.2	0.215	100.0	33.2	0.106
Composite ¼-in. to 0.	Under 1.40	47.8	9.1	0.044	47.8	9.1	0.044
	1.40–1.50	10.8	23.8	0.043	58.6	11.8	0.043
	1.50–1.60	6.5	34.6	0.065	65.1	14.1	0.046
	1.60–1.70	4.3	42.7	0.087	69.4	15.9	0.049
	Over 1.70	30.6	70.6	0.335	100.0	32.6	0.132

[a] Moisture-free basis.

If phosphorus occurs largely in only certain horizons of a coal bed, or if certain areas of a bed are low in phosphorus, a possibility may exist for selective mining, provided, of course, that such mining can be employed economically. Accordingly, some beds of coking coal in the state were channel-sampled in benches over as large an area as possible, to examine the possibilities of selective mining. Sampling in this manner, since each parting or band of impurity was sampled separately, also provides information on how the phosphorus content of the bed would be affected by washing. The analytical procedure used for determining phosphorus is described in *Technical Paper No.* 8, Bureau of Mines.[7]

SPECIFIC-GRAVITY DISTRIBUTION OF PHOSPHORUS IN WILKESON WEST 3 COAL

A 6-ton channel sample for specific-gravity analysis and washing tests was mined from the No. 3 bed on the west dip at Wilkeson, Pierce County, Washington. The channel was cut from hanging wall to footwall and thus contained all of the impurities within the bed. This sampling procedure was adopted because the bed dips about 85° and thus no hand picking at the face can be practiced. The coal was crushed to pass an 1195 Ton-Cap screen (about ¼-in.) preparatory to testing.

Table 1 gives the results of specific-gravity analyses of this coal. A sharp break in phosphorus content at 1.70 sp. gr. is evident. Material heavier than 1.70 sp. gr.

contains about 7 times as much phosphorus as does the coal lighter than 1.70. This coal could be reduced from 0.132 to 0.049 per cent phosphorus by removing the impurity heavier than 1.70 sp. gr. Removing the next lighter fractions, 1.60 to 1.70 and 1.50 to 1.60 sp. gr., would effect little additional reduction in phosphorus content, however.

TABLE 2.—*Specific-gravity Analysis of Coal from Wilkeson East 2 Bed, Crushed through ¼ Inch*

Size	Specific Gravity	Weight, Per Cent	Ash,[a] Per Cent	Phosphorus,[a] Per Cent	Cumulative, Per Cent		
					Weight	Ash[a]	Phosphorus[a]
¼-in. to 20-mesh. Weight, 63.9 per cent.	Under 1.30	21.5	4.9	0.063	21.5	4.9	0.063
	1.30–1.40	38.2	13.4	0.088	59.7	10.3	0.079
	1.40–1.70	24.8	29.7	0.111	84.5	16.0	0.088
	Over 1.70	15.5	67.2	0.856	100.0	24.0	0.253
Under 20-mesh. Weight, 36.1 per cent.	Under 1.30	19.9	3.6	0.058	19.9	3.6	0.058
	1.30–1.40	38.8	11.1	0.074	58.8	8.6	0.069
	1.40–1.70	29.6	26.2	0.072	88.4	14.5	0.076
	Over 1.70	11.6	63.0	0.457	100.0	20.1	0.127
Composite ¼-in. to 0.	Under 1.30	20.9	4.5	0.062	20.9	4.5	0.062
	1.30–1.40	38.5	12.5	0.083	59.4	9.7	0.075
	1.40–1.70	26.6	28.3	0.095	85.9	15.4	0.082
	Over 1.70	14.0	66.0	0.737	100.0	22.6	0.206

[a] Moisture-free basis.

The conclusions based on specific-gravity analysis were verified by a washing test made with a full-sized coal-washing table. A washed coal containing 12.1 per cent ash and 0.043 per cent phosphorus was produced by removing a refuse product containing 59.3 per cent ash and 0.358 per cent phosphorus.

SPECIFIC-GRAVITY DISTRIBUTION OF PHOSPHORUS IN WILKESON EAST 2 COAL

The sample from the east dip of the No. 2 bed in the Bartoy mine at Wilkeson comprised a 35-ton lot obtained for washing and coking tests; it was crushed to pass the 1195 Ton-Cap screen before testing. Table 2 shows the results of specific-gravity analyses of this coal. With this coal also there is a sharp break in the phosphorus content at 1.70 sp. gr. The impurity heavier than this contains almost 9 times as much phosphorus as the coal under 1.70 sp. gr. Removing the 14 per cent of heavy impurity with its 0.737 per cent of phosphorus would reduce the phosphorus content of this coal from 0.206 to 0.082 per cent.

A relationship between phosphorus content and size of coal also is evident (Table 2). The coal finer than 20-mesh contains only half as much phosphorus as that occurring in the coarser size. The difference is attributable almost wholly to the phosphorus contents of the respective fractions over 1.70 sp. gr. The coarse size of heavy impurity contains 0.856 per cent phosphorus, while the fine size contains only 0.457 per cent. Coal of less than 1.70 sp. gr. exhibits little difference in phosphorus content in the two sizes.

This coal is difficult to wash because it contains a large amount of intermediate-density material. In washing it on the table, a washed coal containing 12.0 per cent ash and 0.068 per cent phosphorus resulted, but it was necessary to discard a middling as well as a refuse product.

SPECIFIC-GRAVITY DISTRIBUTION OF PHOSPHORUS IN WILKESON WEST 5 COAL

Table 3 shows the results of a specific-gravity analysis of a 15-ton lot of coal from the west dip of the No. 5 bed in the Wilkeson-Miller mine at Wilkeson crushed to pass the 1195 Ton-Cap screen. This coal differs from those previously described principally in the higher percentage of phosphorus in the lighter fractions. Even the coal lighter than 1.30 sp. gr. has 0.118 per cent phosphorus. The heavy impurity, over 1.70 sp. gr., contains only 2½ times as much phosphorus as does the coal lighter than 1.70. Removing the impurity heavier than 1.70 sp. gr. would reduce the phosphorus content of this coal only from 0.152 to 0.141 per cent.

TABLE 3.—*Specific-gravity Analysis of Coal from Wilkeson West 5 Bed, Crushed through ¼ Inch*

Size	Specific Gravity	Weight, Per Cent	Ash,[a] Per Cent	Phosphorus,[a] Per Cent	Cumulative, Per Cent		
					Weight	Ash[a]	Phosphorus[a]
¼-in. to 20-mesh. Weight, 81.5 per cent.	Under 1.30	74.8	5.1	0.122	74.8	5.1	0.122
	1.30–1.40	16.0	12.9	0.196	90.8	6.5	0.135
	1.40–1.70	5.1	30.7	0.260	95.9	7.8	0.142
	Over 1.70	4.1	72.1	0.350	100.0	10.4	0.150
Under 20-mesh. Weight, 18.5 per cent.	Under 1.30	51.2	4.4	0.094	51.2	4.4	0.094
	1.30–1.40	29.6	10.6	0.177	80.8	6.7	0.124
	1.40–1.70	9.6	25.8	0.263	90.4	8.7	0.134
	Over 1.70	9.6	74.3	0.348	100.0	15.0	0.160
Composite, ¼-in. to 0.	Under 1.30	70.4	5.0	0.118	70.4	6.0	0.118
	1.30–1.40	18.5	12.2	0.191	88.9	6.5	0.133
	1.40–1.70	5.9	29.4	0.261	94.8	7.9	0.141
	Over 1.70	5.2	71.7	0.350	100.0	11.4	0.152

[a] Moisture-free basis.

TABLE 4.—*Specific-gravity Distribution of Phosphorus in Coal from Wilkeson East 4 Bed, Crushed through ¼ Inch*

Sample No.	Specific Gravity	Weight, Per Cent	Ash,[a] Per Cent	Phosphorus,[a] Per Cent
1	Under 1.60	98.8	9.2	0.234
	Over 1.60	1.2	54.0	0.280
	Composite[b]	100.0	9.7	0.235
2	Under 1.60	92.5	9.6	0.293
	Over 1.60	7.5	60.3	0.285
	Composite	100.0	13.4	0.292
3	Under 1.60	97.7	9.9	0.245
	Over 1.60	2.3	52.0	0.147
	Composite	100.0	10.9	0.243
4	Under 1.60	98.2	9.7	0.266
	Over 1.60	1.8	54.9	0.180
	Composite	100.0	10.5	0.264
5	Under 1.60	95.2	9.4	0.195
	Over 1.60	4.8	50.3	0.255
	Composite	100.0	11.4	0.198
6	Under 1.60	94.1	11.2	0.202
	Over 1.60	5.9	55.2	0.085
	Composite	100.0	13.8	0.195

[a] Moisture-free basis.
[b] Calculated from the two specific-gravity fractions.

In a cleaning test made with the coal-washing table, a washed coal containing 7.1 per cent ash and 0.131 per cent phosphorus was produced by rejecting a refuse product containing 53.5 per cent ash and 0.343 per cent phosphorus.

SPECIFIC-GRAVITY DISTRIBUTION OF PHOSPHORUS IN WILKESON EAST 4 COAL

Channel samples were cut from the east dip of the No. 4 bed in the Wilkeson-Wingate, Dependable, and Apex mines at Wilkeson, the sampling points covering a distance along the strike of the bed of about 1½ miles. All impurities within the bed were included in the samples, each of which weighed about 20 lb. The samples were crushed to pass a ¼-in. screen and then tested separately at 1.60 sp. gr. (Table 4).

It is clearly evident that a coal containing less than about 0.2 per cent phosphorus could not be produced from this area of the bed by a separation at 1.60 sp. gr. The phosphorus appears to be about equally distributed between the coal and impurity. In fact, in four out of the six samples the impurity actually contains less phosphorus than the float coal. Consequently, washing probably would concentrate rather than eliminate the phosphorus; that is, the washed coal would contain a higher percentage of phosphorus than the raw coal from which it was derived.

DISTRIBUTION OF PHOSPHORUS IN BEDS OF THE ROSLYN-CLE ELUM COAL FIELD

In studying the occurrence of phosphorus in beds of the Roslyn-Cle Elum coal field in Kittitas County, channel sampling of the beds by separate benches and impurity partings at a number of points was used instead of specific-gravity analyses. It was hoped that an area of low-phosphorus coal might be outlined by this procedure. In addition to providing data on the areal distribution of phosphorus, the sampling procedure furnished information on the extent to which phosphorus content could be reduced by washing, for the impurities in the bed were analyzed separately.

TABLE 5.—*Distribution of Phosphorus in Various Components of the Roslyn Coal Bed*

Section	Number of Samples	Average[a] Phosphorus, Per Cent	Distribution of Samples by Phosphorus Content						
			Under 0.05	0.05 to 0.1	0.1 to 0.2	0.2 to 0.3	0.3 to 0.5	0.5 to 0.7	Over 0.7
Roof, sandy shale	1	0.087	0	1					
Cap rock, shale	6	0.144	0	2	3	1			
Coal, upper bench	15	0.185	0	1	8	5	1		
Middle sandstone, bony coal, and shale	11	0.532	0	0	2	2	1	4	2
Coal, lower bench	15	0.162	0	2	11	2			
Bottom shale and bony coal	9	0.050	4	5					
Floor, sandy shale	1	0.041	1						

[a] Moisture-free basis.

The Roslyn bed was sampled at 15 points throughout an area about 4 miles long and 1 mile wide in the Nos. 3, 5, and 9 mines of the Northwestern Improvement Co. No significant areal variation in phosphorus content could be detected from these samples; that is, the variation in phosphorus content between individual samples in any one locality was as great as the difference between the average content for the locality and that for the area as a whole. Although some areal variation in phosphorus content was indicated by the 10 samples taken in the No. 3 and No. 6 beds, the number of sampling points was too small to yield conclusive evidence.

Roslyn Bed

While no area of low-phosphorus coal was discovered, some interesting information was obtained with respect to the vertical distribution of phosphorus in the coal beds. Table 5 summarizes the results of samples taken at 15 locations in the Roslyn bed, a section of which is shown in Fig. 1.

The phosphorus content of the coal in the bed averages about 0.175 per cent, and is slightly higher in the upper bench than in the lower. Either bench may vary from less than 0.1 to as much as 0.3 per cent in the area studied. The impurities associated with the coal vary in phosphorus content primarily with their distance, either up or down, from the middle of the bed. The characteristic middle parting, composed of sandstone, bony coal, and shale, averages over 0.5 per cent phosphorus and at some locations contained over 0.7 per cent. The shale cap rock overlying the coal contains somewhat less phosphorus than either bench of coal, and the bottom impurity, floor, and roof all are less than 0.1 per cent in phosphorus content.

The gradation in phosphorus content from high in the middle of the bed to low in the roof and floor prompted additional sampling to determine whether a corresponding gradation occurred within the coal benches themselves; that is, whether the coal adjacent to the middle parting contained more phosphorus than that closer to the walls of the bed. At two locations in the No. 9 mine the upper and lower portions of coal were sampled in 2-in. and 6-in. arbitrary benches, and the three components of the middle parting were sampled separately. Fig. 1 illustrates the results obtained at one of these sampling points. In the upper bench, the phosphorus ranges from 0.943 per cent in the 2-in. increment of coal adjacent to the middle parting to 0.225 per cent in the coal next to the cap rock. In the lower bench, however, no gradation is evident; the 2-in. increment containing the highest percentage of phosphorus in the entire section lies adjacent to the 2-in. increment having the lowest phosphorus content of any coal in the section. At the second sampling location, no significant gradation in phosphorus content occurred in the coal.

SECTION	INCHES	ASH, PER CENT	PHOSPHORUS, PER CENT
ROOF, SANDY SHALE		94.1	0.165
CAP ROCK, BONE	21	50.7	0.098
COAL	7	12.2	0.225
COAL	6	20.9	0.446
COAL	6	14.3	0.245
COAL	2	20.2	0.681
COAL	2	26.6	0.943
SANDY SHALE	1½	87.2	0.430
COAL, BONY	1	35.4	1.080
SHALE	1½	87.7	0.295
COAL	2	22.0	1.210
COAL	2	17.7	0.110
COAL	6	13.1	0.210
COAL	6	10.9	0.169
COAL	6	11.4	0.176
COAL AND SHALE	3½	55.6	0.060
FLOOR, SANDY SHALE		92.4	0.097

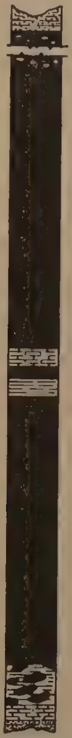

FIG. 1. SECTION OF ROSYLN BED, SHOWING PHOSPHORUS CONTENT OF IMPURITIES AND OF 2-INCH AND 6-INCH INCREMENTS OF COAL.

Perhaps the most striking feature of the data in Fig. 1 is the pronouced variation in the phosphorus content of the various portions of the bed. Some of the impurities are low in phosphorus content, others are unusually high; similarly, some of the coal contains only a moderate percentage of phosphorus, while some of it contains a high percentage.

A series of samples of washed coal and refuse collected at the cleaning plant while about 1000 tons of coal from the No. 3 mine was being washed gave the figures of Table 6.

TABLE 6.—*Phosphorus Content of Roslyn Washery Products*[a]

Size of Coal, In.	Ash, Per Cent	Phosphorus, Per Cent
Egg, 3 to 1⅝	11.8	0.141
Nut, 1⅝ to ¾	13.0	0.168
Pea, ¾ to ¼	11.4	0.184
Slack, ¼ to 0, dry[b]	15.5	0.140
Slack, ¼ to 0, wet[c]	9.9	0.145
Refuse, all sizes	61.5	0.286

[a] Moisture-free basis.
[b] Cleaned on pneumatic table.
[c] Cleaned on wet table; coal recovered from primary jig and pneumatic-table refuse.

The exact reduction in phosphorus content obtained in washing cannot be determined because the raw coal was not sampled. However, the comparatively small difference between the percentage of phosphorus in the washed coal and that in the refuse indicates that the reduction was not great. Some variation in phosphorus content with size of coal is evident, and this variation does not appear to be related to ash content.

No. 3 and No. 6 Beds

The No. 3 bed was sampled in the Roslyn No. 3 and Jonesville No. 3 mines, and the No. 6 bed was sampled in the Roslyn-Cascade No. 4 and Roslyn-Fuel Co. No. 3 mines. At each sampling point in these beds all of the coal in the bed was combined in one sample and all of the impurities were placed in another. Thus the samples show roughly the phosphorus content of the clean coal that could be produced from the bed and the character of the refuse that could be removed in a washing operation. Table 7 presents the results of analyses on these samples. The two beds are similar in that the phosphorus content of both varies considerably from place to place and is high. In both beds the impurities contain less phosphorus than the coal; consequently the phosphorus content of neither coal could be reduced by washing.

PETROGRAPHIC EXAMINATION OF HIGH-PHOSPHORUS SHALE OF ROSLYN BED

In an effort to determine whether the phosphorus in the Roslyn bed occurred in mineral form, a sample of the shale high in phosphorus from the middle parting of the bed was separated into a number of density fractions for petrographic examination. Shale rather than coal was chosen for this work because it was abnormally high in phosphorus content, and thus, it was believed, was more likely to disclose the presence of phosphorus minerals. Table 8 shows that the phosphorus content of the individual density fractions increased with increase in density, reaching a maximum of over 10 per cent in the material heavier than 2.7 specific gravity.

TABLE 7.—*Phosphorus Content of Coal and Impurities in Nos. 3 and 6 Beds*

Bed No.	Sampling Point	Phosphorus, Per Cent[a]	
		Coal	Impurity
3	1	0.190	0.040
	2	0.310	0.117
	3	0.263	0.043
	4	0.291	0.216
	5	0.353	0.420
6	1	0.175	0.159
	2	0.106	0.065
	3	0.263	0.090
	4	0.355	0.153

[a] Moisture-free basis.

TABLE 8.—*Specific-gravity Analysis of High-phosphorus Shale from Middle Parting of Roslyn Bed*

Specific Gravity	Weight, Per Cent	Phosphorus,[a] Per Cent	Cumulative, Per Cent	
			Weight	Phosphorus[a]
Under 1.6	12.0	0.57	12.0	0.57
1.6–2.0	8.0	1.98	20.0	1.13
2.0–2.2	29.2	1.94	49.2	1.61
2.2–2.3	17.4	1.59	66.6	1.60
2.3–2.4	21.5	2.21	88.1	1.75
2.4–2.5	7.3	4.56	95.4	1.97
2.5–2.6	1.2	5.44	96.6	2.01
2.6–2.7	3.3	6.50	99.9	2.16
Over 2.7	0.1	10.37	100.0	2.17

[a] Moisture-free basis.

The phosphorus minerals evansite ($3Al_2O_3.P_2O_5.18H_2O$) and wavellite ($3Al_2O_3.2P_2O_5.13H_2O$) were identified in the petrographic examination, but the amount in which they occurred accounts for only about 12 per cent of the total phosphorus in the shale. The form in which the remaining 88 per cent of the phosphorus occurred could not be determined petrographically.

CONCLUSIONS

The elimination of phosphorus by coal-cleaning methods depends entirely upon its mode of occurrence, therefore each coal bed offers an individual problem in phosphorus removal. The phosphorus content of two beds studied, the east No. 2 and the west No. 3 at Wilkeson, Pierce County, is reduced greatly by the washing procedure required to produce coal of satisfactory ash content. In the other five beds examined the phosphorus is associated with the clean coal rather than with the impurity, hence cannot be eliminated by ordinary coal-cleaning methods.

The lowest phosphorus content obtained with the coals studied was 0.043 per cent; still lower figures might be obtained with other beds of coking coal formerly worked but not available for sampling now—for example, those in the Carbonado, Fairfax, and Montezuma areas.

ACKNOWLEDGMENTS

The authors wish to acknowledge the interest and helpful criticism extended by Milnor Roberts, Dean, and Joseph Daniels, Professor of Mining Engineering, both of the College of Mines of the University of Washington. The spirit of cooperation in which the coal operators in the Roslyn and Wilkeson areas assisted in obtaining samples of their mines is especially appreciated. The authors are indebted also to K. G. Skinner, Assistant Chemical Engineer of the Northwest Experiment Station of the Bureau of Mines, for the petrographic examination of shale from the Roslyn coal bed.

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DISCUSSION

(See section on Phosphorus in paper by Davis and Griffen, page 54, this volume, for further discussion of this subject.)

G. H. CADY,* Urbana, Ill.—The failure of the authors to investigate the possibility of some relationship between the phosphorus content of the Washington coals and the banded ingredients leaves one possibility unexplored. The writer has not sufficient familiarity with bituminous coals produced in the state of Washington to know whether or not the usual banded constitution of bituminous coals characterizes the particular coals studied, but he suspects that it does. If so, the various petrographic ingredients are probably separable.

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* Illinois State Geological Survey.

The exploration of this possibility would have to be conducted by using fractions separated at points different from those used for both gravity and size, in order to establish the relationship of phosphorus content to petrographic components. Separation even at 1.30 to 1.40 sp. gr. (Tables 1, 2, and 3) is still too high to obtain a differentiation of the physical ingredients. Hence, although a coal as light or lighter than this may contain a relatively low phosphorus content, such a coal may be a low-ash variety of any one of the ingredients. With Illinois coal, a fairly clear-cut separation of vitrain from other types of coal can be obtained at 1.22 or even 1.25 sp. gravity.

The separation made at 1.60 sp. gr. shown in Table 4 is altogether too high to show whether the coal or the mineral matter contains the phosphorus and whether or not the phosphorus is concentrated in certain parts of the coal. The statement that a coal containing less than 0.2 per cent phosphorus cannot be produced from this area of the bed by a separation at 1.60 sp. gr. is apparently descriptive of the facts, but if No. 4 bed contains coal such as is apparently found in No. 5 bed (Table 3), separation at 1.40 should produce the coal desired.

The profile of the Roslyn bed (Fig. 1) divides the coal bed into blocks irrespective of their physical constitution. The very fact that large variations in the phosphorus content exist irrespective of ash content is suggestive of a source of variation not recognized in the investigation. Inasmuch as separation of banded bituminous coal into petrographic types of coal material is usually not difficult; it seems possible that such separation might have been undertaken in this case.

The work of the Illinois Geological Survey has never included a systematic investigation of the phosphorus content of Illinois coals. We do not know how the phosphorus content is distributed, geographically or by bed, and much less with respect to the banded ingredients. Hence, our investigations provide no basis for generalizations in regard to the relation of phosphorus content to the banded ingredient. Nevertheless, since the ingredients possess rather definite characteristics in other particulars, some consideration of this possibility of relationship merits consideration.

Stutzer and Noe (Geology of Coal, p. 21) cite Frech, who calls attention to the remarkable correlation between the high phosphorus content and the high spore content of certain French coal of the cannel type. May not a similar relationship exist with respect to the coals of the state of Washington?

H. H. Lowry,* Pittsburgh, Pa.—It has been stated frequently that all the phosphorus in a coal remains in the coke and all the phosphorus in the coke is completely transferred to the iron in the blast furnace. Convincing experimental evidence of the correctness of these statements has not been found. Phosphorus is chemically similar to nitrogen, which is partly recovered in the tar and partly in the gas and liquor on carbonization. Combustion of coke at the blast-furnace tuyeres might be expected to produce phosphorus pentoxide, which would be picked up by the lime and enter the slag. Because of the very small amounts of phosphorus in most coals, it may be that traditional acceptance of the statements made in the first sentence might be excused as a simplifying assumption of little consequence. However, with the high-phosphorus coals described by the authors, the truth or falsity of both statements becomes of great significance. Do the authors have data, for instance, to show that all the phosphorus of the Washington coals remains in the coke made from them?

M. R. Geer (author's reply).—As Dr. Cady has observed, correlation of the phosphorus content of the coals with their petrographic composition might have yielded interesting information on the association of the phosphorus. However, the present work had the specific objective of determining whether or not the phosphorus was associated with the dense impurities that are discarded in the ordinary coal-washing operation. Separations at the low specific gravities mentioned by Dr. Cady—namely, 1.22 or 1.25—would not be economically feasible with these coals, or, for that matter, with most other coals. The coals examined probably would not be washed at a specific gravity of less than 1.50 as a minimum. Thus, while a low-phosphorus product might be obtained at these low specific gravities, it could not be duplicated by plant operation.

Information on the petrographic composition of two Washington coals from the Wilkeson

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* Coal Research Laboratory, Carnegie Institute of Technology.

TABLE 9.—*Comparison of Phosphorus in Coal and Resultant Coke*

Sample No.	Ash, Per Cent[a]		Phosphorus, Per Cent[a]		Calculated Phosphorus in Coke, Per Cent[a,b]
	Coal	Coke	Coal	Coke	
1	11.9	15.0	0.059	0.075	0.074
2	11.9	14.6	0.054	0.054	0.066
3	12.0	14.7	0.056	0.064	0.069
4	10.8	13.1	0.050	0.069	0.061
5	12.4	16.2	0.105	0.143	0.137
6	13.3	16.4	0.099	0.139	0.122
7	9.4	12.3	0.182	0.246	0.238
8	11.5	14.2	0.062	0.083	0.077
9	11.4	14.1	0.053	0.062	0.066
10	7.4	9.8	0.050	0.078	0.066
11	13.5	16.0	0.054	0.068	0.064
12	6.2	8.8	0.021	0.039	0.030
13	9.2	15.9	0.060	0.108	0.104
14	14.9	17.1	0.120	0.190	0.138
15	10.8	11.9	0.057	0.064	0.063
16	11.9	15.7	0.067	0.094	0.088
17	7.1	10.7	0.145	0.230	0.219
18	10.6	16.0	0.179	0.228	0.270
19	8.4	11.3	0.044	0.070	0.059
20	7.3	10.1	0.065	0.095	0.090
21	7.8	10.8	0.086	0.125	0.119
22	8.0	11.2	0.090	0.138	0.126
23	6.6	9.1	0.080	0.107	0.110

[a] Moisture-free basis.

[b] Phosphorus in coal $\times \frac{\text{coke ash}}{\text{coal ash}}$.

area is given in *Technical Paper* 649 of the Bureau of Mines.

The present investigation was not concerned with Dr. Lowry's question as to the behavior of phosphorus during the carbonization of coal; two other reports of work by the Bureau of Mines, however, contain some pertinent information on this point.* Table 9 shows the percentages of ash and phosphorus in 23 coals and the corresponding cokes. These data represent coke-plant operation rather than laboratory tests and hence are subject to the errors of sampling that necessarily characterize such information. No evidence of any loss of phosphorus during carbonization is shown by these figures, for with most of the samples the amount of phosphorus in the coke equals or exceeds that calculated by assuming that all of the phosphorus in the coal is retained in the coke.

It should be pointed out in this connection that the procedure for determining phosphorus described in *Technical Paper* 8 of the Bureau of Mines assumes that no phosphorus is lost during combustion, for the determination is made on the ash of the sample only.

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* H. F. Yancey, R. E. Zane, R. W. Fatzinger and J. A. Key: Physical and Chemical Properties of Cokes Made or Used in Washington. U. S. Bur. Mines *Tech. Paper* 597 (1939).

H. F. Yancey, J. Daniels, E. R. McMillan and M. R. Geer: Byproduct Coke Oven Tests of Washington Coals. U. S. Bur. Mines *R.I.* (in press).

The Washing of Pittsburgh Coking Coals and Results Obtained on Blast Furnaces

By C. D. King,* Member A.I.M.E.

(Cleveland Meeting, April 1943)

The key to maximum production of ingots for the war effort is maximum production of pig iron. For any given furnace and ore, the most important single influence on blast-furnace production is the quality of the furnace coke.

It is generally recognized that undoubtedly deterioration of Pittsburgh coking coals will be accelerated by the depletion of some of the better grades of coal; the extension of mine mechanization, with attendant degradation of coal quality; and the opening of new coal fields, some of which are known to be fairly high in sulphur. All of these factors, contributing as they do to higher ash and sulphur, and not infrequently to inferior quality of coke, made it desirable that an extensive study be undertaken to determine the effect of washing some coals that closely approximate the average chemical composition of coking coals to be expected in the future by the subsidiaries of the United States Steel Corporation in the Pittsburgh district. It was deemed advisable to study the improvement obtained by washing such coals and to determine the need for supplementing the existing coal washery at Clairton to assure continued maximum production of pig iron from presently available furnace stacks. The existence of the coal washer at Clairton made it possible to conduct tests on coals used in raw form and on similar coals after washing.

This study covers an investigation of the quality of furnace coke as affected by washing coking coals, and is confined to the use of 100 per cent high-volatile Pittsburgh seam coals from Fayette and Greene Counties in the manufacture of furnace coke. It covers the improvement in chemical and physical properties as well as the degree of uniformity in coke obtained by washing these coals, and includes the effect on blast-furnace production and practice. The test period involved is four months; the plants, Clairton by-product coke plant, and the Carrie No. 1 and No. 2 blast furnaces at Rankin, Pa., Carnegie-Illinois Steel Corporation.

CLAIRTON BY-PRODUCT COKE PLANT AND COAL WASHER

The Clairton by-product coke plant carbonizes 30,800 net tons of coal daily in 22 batteries containing 1482 ovens. Three oven sizes are used, commonly known as wide, medium, and narrow types, recognizing the difference in oven width. These ovens also differ in height and length.

The present coal washer has a daily capacity of approximately 14,000 tons, leaving 16,800 tons to be carbonized in raw form. Washed coal is used on all the narrow ovens and on three of the six batteries with ovens of medium width, raw coals being used on all the remaining ovens. The customary arrangement is

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shown in Table 1, which also includes various oven dimensions.

The use of raw coal for approximately 55 per cent of the total Clairton output has in the past yielded satisfactory results, since relatively low-sulphur and low-ash coals were available for use in the raw state. Palmer and Gates coal, normally coals of moderately low sulphur-ash, have been carbonized in this form, whereas Colonial and Ronco, which are generally inferior, have been washed. The usual combination of Colonial and Ronco mixtures is 70 and 30 per cent, respectively, or in accordance with the mining schedules for these coals. In addition, the allocation of washed coals to some batteries and raw coals to others also recognized the location of the washer, which serves coal bunkers for only seven of the batteries.

TABLE 1.—*Oven Dimensions and Normal Allocation of Raw and Washed Coals, Clairton By-product Coke Plant*

Type of oven	Koppers	Koppers-Becker Combination		
Number of batteries of each	12	6		4
Battery number or other designation	Nos. 1–12, incl. (wide)	Nos. 13–18, incl. (medium)		Nos. 19–22, incl. (narrow)
Number of ovens per battery	64	61		87
Number of ovens	768	366		348
Oven dimensions:				
Width: Coke side	19½″	18″		17½″
Width: Pusher side	17″	16″		15½″
Width: Average	18¼″	17″		16½″
Length between doors	36′ 6½″	40′ 4½″		42′ 6″
Height: Floor to roof	9′ 10⅝″	11′ 8⅛″		14′ 0″
Height: Coal charge	8′ 10⅝″	10′ 8⅛″		13′ 0″
Capacities:				
Cubic feet	500	616		759
Coal charge per oven, net tons	13.24	16.70		20.63
Coal charged per unit per day, net tons	12,700	8,200		9,900
Net coking time, hours	18.6	17.7		16.8
		Batt. 13–15	Batt. 16–18	
Coal carbonized	Raw Palmer	Raw Gates	Washed Colonial-Ronco	Washed Colonial-Ronco
Coal analysis:				
Ash, per cent	8.70	8.25	8.91[a]	8.91[a]
Sulphur, per cent	1.00	1.00	1.36[a]	1.36[a]

[a] Prior to washing.

Coking coals for the Clairton plant are shipped down the Monongahela River by barge, each barge holding about 900 tons. The coal is unloaded from barges into bins by clamshell buckets at four coal hoists, from which it passes over screens, the coarse coal being fed to roll crushers set at 2-in. openings and the fine coal bypassing the crusher. The mixed crushed and fine coal is carried by conveying belts to the various coal bunkers serving the batteries, each bunker having a capacity of 4000 tons of coal. From the bunkers it is handled in the usual manner by larry cars to the ovens.

The Clairton by-product coke plant is unique in its use of 100 per cent high-volatile, Pittsburgh seam coals and also the fact that the coal is not pulverized prior to carbonization. Long experience at Clairton has shown that under existing conditions it is desirable to charge unpulverized coal to obtain maximum bulk density and, accordingly, maximum coke output.

By avoiding the fine crushing of coal, maximum bulk density is similarly obtained when employing washed coal. This practice also possesses the additional advantage of eliminating a considerable amount of slate as large pieces by the coarse coal launders and thereby avoids the more difficult fine launder elimination of this proportion as fine slate particles. The present coal washer, installed in 1931, is the Rheolaveur type. The washer was built on the north end of the plant in order to utilize the capacity of the larger batteries as well as to permit the use of coal-handling facilities already available. After leaving the roll crushers at the coal hoists, the mixed coarse and fine coal is conveyed to three double-compartment blending bins, with a total capacity of 3000 tons. This arrangement permits the handling of three types of coal, Colonial, Ronco, and any other coal received at Clairton if the first two are not available in quantity to utilize the full capacity of the washer. Variable-speed belt conveyors at the bottoms of the bins allow for adjustment in desired coal mixtures fed to the washing plant. Coals are screened to two sizes, plus and minus 5/16-in., prior to washing.

TABLE 2.—*Average of Control Data for Coal-washery Float-and-sink Test during the Four-months Washing Test, Clairton By-product Coke Plant*

Coal	Size	Dry, Per Cent	Sieve, Per Cent	Float 1.55 Sp. Gr., Per Cent			Sink 1.55 Sp. Gr., Per Cent			Composite[a]	
				Total	Ash	Sulphur	Total	Ash	Sulphur	Ash	Sulphur
Raw......	+5/16″		63.43	95.44	7.05	1.04	4.56	61.25	5.05	9.52	1.22
	5/16″–20m.		26.09	96.15	5.72	1.07	3.85	59.67	7.49	7.80	1.32
	–20m.		10.48	93.49	6.11	1.06	6.51	60.79	11.93	9.67	1.77
	Over-all		100.00	95.42	6.60	1.05	4.58	60.83	6.61	9.09	1.31
Washed....	+5/16″		60.82	99.59	7.08	1.04	0.41	40.43	3.65	7.22	1.05
	5/16″–20m.		25.14	99.24	5.86	1.04	0.76	36.76	4.36	6.09	1.06
	–20m.		14.04	96.70	5.83	1.04	3.30	53.68	11.93	7.41	1.40
	Over-all	96.63	100.00	99.10	6.60	1.04	0.90	46.45	8.04	6.96	1.10
Refuse	+5/16″		47.30	5.01	17.66	1.78	94.99	65.59	5.48	63.18	5.29
	5/16″–20m.		40.54	10.37	11.87	1.70	89.63	64.86	8.24	59.36	7.56
	–20m.		12.16	26.64	8.91	1.32	73.36	71.21	15.22	54.61	11.52
	Over-all	3.37	100.00	9.79	12.29	1.60	90.21	65.85	7.56	60.60	6.97

[a] Calculated from float-and-sink-test data.

The washery was designed to yield a three-product separation; i.e., washed coal, middlings for boiler fuel, and refuse. Experience with coals received to date at Clairton has shown that under existing conditions it has not been economical to remove bone coal as middlings because of the small percentage present in raw coal. However, the provision for a middling product may prove highly desirable as insurance against adverse changes in sulphur and ash content of future coals.

COKE-PLANT TESTS

The study of the effect of washing Colonial and Ronco coals extended over a period of four months. The comparison was based on carbonizing washed Colonial-Ronco coals in the same battery used for coking Colonial-Ronco raw coals. This arrangement was considered advisable in order to eliminate any coke-plant variable other than the type of coals used. Since Colonial-Ronco washed coal is customarily carbonized in batteries 19-22, and because these batteries were also provided with a spare coal bunker, arrangements were made to use this spare for storing Colonial-Ronco raw coals. Accordingly, 30 ovens in batteries 19 and 20 were charged with Colonial-Ronco unwashed coal and the remainder of the ovens in the same batteries were charged with Colonial-Ronco washed coal. The mixture of Colonial-Ronco coal was approximately 70 per cent of the former and 30 per cent of the latter.

Comparison of Analyses of Coals Used and Coke Produced

The high efficiency of the coal washer is indicated by the fact that in a two-product separation 96.63 per cent of the initial coal was recovered as a washed product; the refuse, or bank loss, being 3.37 per cent.

The refuse contained 4.91 per cent sulphur and 62.71 per cent ash. The plus 5/16-in. washed coal at 1.55 sp. gr. contained only 0.41 per cent sink and the 5/16 in. to 20 mesh, 0.76 per cent. The refuse showed only 9.79 per cent float coal with an ash content of 12.29 per cent. Of the initial coal, less than 0.3 per cent was lost in the refuse. Detailed float-and-sink test data are shown in Table 2.

TABLE 3.—*Composite of Four-months Test Results Showing Effect of Coal Washing on Coal and Coke Analyses*
PER CENT

Proximate Analyses, Dry Basis	Unwashed Coals	Washed Coals	Effect of Washing
Moisture	3.93	7.27	+3.34
Sulphur	1.36	1.11	−0.25
Ash	8.91	7.12	−1.79
Volatile matter	32.49	32.97	+0.48
Fixed carbon	58.60	59.91	+1.31

Proximate Analyses, Dry Basis	Furnace Coke from		Effect of Washing
	Unwashed Coals	Washed Coals	
Moisture	2.95	2.88	
Sulphur	0.98	0.88	−0.10
Ash	11.87	10.36	−1.51
Volatile matter	0.30	0.30	
Fixed carbon	87.83	89.34	+1.51
Available carbon (2.5 per cent moisture basis)	83.02	84.58	+1.56

The results obtained by washing Colonial-Ronco coals for the four-months test period are shown in Table 3. The ash in the coal was reduced by 1.79 per cent and the sulphur 0.25 per cent; the fixed carbon increased by 1.31 per cent and the moisture 3.34 per cent. The sulphur content of 1.36 per cent in the initial coal was made up of 0.80 per cent organic sulphur, 0.54 per cent pyritic, and 0.02 per cent sulphate. Washing these coals resulted in the elimination of approximately 45 per cent of the pyritic sulphur. The furnace coke produced from washed coals was lower in ash than the coke from unwashed coals by 1.51 per cent, and the sulphur 0.10 per cent. The fixed carbon was increased 1.51 per cent. Detailed data showing the results for each of the four months are shown in Table 5.

The reduction of 1.51 per cent ash in furnace coke from washed coals does not fully account for the reduction of 1.71 per cent ash in coal by washing. The remainder of this ash difference will be found in the coke breeze produced from the respective coals, being approximately 20 per cent ash in coke breeze from unwashed coals and only 12 per cent in coke breeze from washed coals.

Comparison of Physical Properties of Coke Produced from Washed and Unwashed Coals

In addition to the beneficial effects obtained with respect to reduction of sulphur and ash in coal and coke by washing Colonial-Ronco coals, a marked improvement in physical characteristics of the furnace coke was also derived. This is attributed to the greater tendency of coke made from raw coal to cross-fracture because of slate inclusions. Slate in coal over 3/8 in. is relatively easy to eliminate, but its reduction is particularly desirable in sizes ranging from 3/8-in. to 20-mesh, since the number of incipient fractures caused by slate in coal is far greater in the smaller size than in the larger for equal percentages

TABLE 4.—*Composite of Four-months Tests Showing Effect of Coal Washing on Physical Properties of Coke*
PER CENT

Test	Furnace Coke from		Effect of Washing
	Unwashed Coals	Washed Coals	
Sieve Test			
On 3 in.	14.3	15.8	+ 1.5
Total on 2 in.	60.2	63.8	+ 3.6
Through 1 in.	2.4	1.7	− 0.7
Tumbler Test			
Strength	65.3	72.0	+ 6.7
Hardness	73.8	76.2	+ 2.4
Brittleness	55.9	52.0	− 3.9
Fuel value	73.5	86.5	+13.0
Weight per cu. ft. (2.5 per cent moisture basis), lb.	31.61	31.41	− 0.20

TABLE 5.—*Detailed Analyses of Washed and Unwashed Coals Used and Coke Produced, Clairton By-product Coke Plant*

PER CENT

Proximate Analyses, Dry Basis	Unwashed Coals						Washed Coals						Over-all Average, July, August, October and November		
	July	August	Average	Oct.	Nov.	Average	July	August	Average	Oct.	Nov.	Average	Un-washed (Raw) Coals	Washed Coals	Difference Washed versus Raw
Moisture	4.05	4.17	4.11	3.80	3.69	3.75	7.29	7.18	7.24	7.36	7.26	7.31	3.93	7.27	+ 3.34
Sulphur	1.36	1.37	1.37	1.35	1.36	1.35	1.10	1.11	1.11	1.12	1.09	1.11	1.36	1.11	− 0.25
Ash	8.87	8.94	8.91	9.00	8.80	8.91	7.09	7.14	7.12	7.13	7.12	7.13	8.91	7.12	− 1.79
Volatile matter	32.53	32.46	32.50	32.46	32.51	32.48	33.00	32.97	32.99	32.94	32.96	32.95	32.49	32.97	+ 0.48
Fixed carbon	58.62	58.60	58.59	58.54	58.69	58.61	59.91	59.89	59.89	59.93	59.92	59.92	58.60	59.91	+ 1.31
Proximate Analyses, Dry Basis	Furnace Coke Produced from Unwashed (Raw) Coals						Furnace Coke Produced from Washed Coals						Coke from:		Difference Washed versus Raw
	July	Au-gust	Aver-age	Oct.	Nov.	Aver-age	July	Au-gust	Aver-age	Oct.	Nov.	Aver-age	Un-washed Coals	Wash-ed Coals	
Moisture	3.02	2.99	3.01	2.85	2.96	2.90	2.93	2.87	2.90	2.84	2.88	2.86	2.95	2.88	
Sulphur	0.98	0.99	0.99	0.98	0.97	0.98	0.89	0.88	0.89	0.89	0.87	0.88	0.98	0.88	− 0.10
Ash	11.72	12.05	11.89	11.85	11.84	11.85	10.42	10.38	10.40	10.38	10.27	10.33	11.87	10.36	− 1.51
Volatile matter	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	
Fixed carbon	87.98	87.64	87.81	87.85	87.86	87.85	89.28	89.32	89.30	89.32	89.43	89.37	87.83	89.34	+ 1.51
Total carbon	85.30	84.95	85.13	85.17	85.19	85.17	86.68	86.74	86.71	86.73	86.86	86.79	85.15	86.75	+ 1.60
Iron in coke as Fe	1.08	1.01	1.05	0.93	0.95	0.94	0.87	0.87	0.87	0.79	0.75	0.77	1.00	0.82	− 0.18
Phosphorus in coke as P	0.014	0.013	0.013	0.013	0.012	0.013	0.013	0.012	0.012	0.011	0.011	0.011	0.013	0.011	− 0.002
Ultimate Analyses, Dry Basis															
Carbon	85.30	84.95	85.13	85.17	85.19	85.17	86.68	86.74	86.71	86.73	86.86	86.79	85.15	86.75	+ 1.60
Hydrogen	0.36	0.35	0.36	0.35	0.36	0.35	0.35	0.35	0.35	0.35	0.36	0.35	0.35	0.35	
Nitrogen	0.87	0.87	0.87	0.86	0.87	0.86	0.85	0.88	0.87	0.88	0.87	0.88	0.86	0.88	+ 0.02
Sulphur	0.98	0.99	0.99	0.98	0.97	0.98	0.89	0.88	0.89	0.89	0.87	0.88	0.98	0.88	− 0.10
Chlorine	0.14	0.13	0.14	0.13	0.13	0.13	0.13	0.14	0.14	0.14	0.13	0.14	0.14	0.14	
Oxygen	0.63	0.66	0.62	0.66	0.64	0.66	0.68	0.63	0.64	0.63	0.64	0.63	0.65	0.64	− 0.01
Ash	11.72	12.05	11.89	11.85	11.84	11.85	10.42	10.38	10.40	10.38	10.27	10.33	11.87	10.36	− 1.51
Available carbon (2.5 per cent moisture basis)	83.17	82.83	83.00	83.04	83.06	83.04	84.51	84.57	84.54	84.56	84.69	84.62	83.02	84.58	+ 1.56

Analyses of Ash																			
Silica	51.76	53.80	52.78	6.28[a]	54.14	53.46	53.83	6.38[a]	52.26	54.18	53.22	5.53[a]	53.83	53.20	53.54	5.53[a]	53.30	53.38	+ 0.08
Iron oxide	13.18	11.96	12.57	1.49	11.16	11.50	11.31	1.34	11.96	11.96	11.96	1.24	10.86	10.42	10.66	1.10	11.94	11.31	− 0.63
Aluminum oxide	29.16	28.74	28.95	3.44	29.11	29.35	29.23	3.46	30.33	28.37	29.35	3.05	29.91	30.75	30.29	3.12	29.09	29.82	+ 0.73
Manganous oxide	0.10	0.09	0.10	0.01	0.10	0.12	0.11	0.01	0.11	0.14	0.13	0.01	0.11	0.12	0.11	0.01	0.11	0.12	+ 0.01
Calcium oxide	2.64	2.69	2.67	0.32	2.37	2.33	2.35	0.28	2.24	2.24	2.24	0.23	2.29	2.11	2.21	0.29	2.56	2.23	− 0.33
Magnesium oxide	0.79	0.80	0.80	0.10	0.78	0.77	0.78	0.09	0.75	0.76	0.76	0.08	0.75	0.71	0.73	0.08	0.79	0.75	− 0.04
Phosphorus pentoxide	0.27	0.25	0.26	0.031	0.26	0.23	0.25	0.030	0.29	0.27	0.28	0.029	0.25	0.24	0.25	0.026	0.25	0.27	+ 0.02
Physical Properties																			
Sieve test:																			
On 3 in.	14.2	14.4	14.3		14.1	14.6	14.3		15.9	15.7	15.8		15.7	15.7	15.7		14.3	15.8	+ 1.5
Total on 2 in.	60.8	59.9	60.4		59.9	59.9	59.9		63.6	63.7	63.6		64.1	64.2	64.1		60.2	63.8	+ 3.6
Through 1 in.	2.3	2.4	2.4		2.4	2.6	2.5		1.7	1.7	1.7		1.7	1.7	1.7		2.4	1.7	− 0.7
Tumbler test:																			
Strength	65.8	65.9	65.8		65.4	64.2	64.9		72.9	70.3	71.6		72.4	72.4	72.4		65.3	72.0	+ 6.7
Hardness	74.0	74.1	74.0		73.9	73.3	73.6		76.0	76.1	76.0		76.3	76.3	76.3		73.8	76.2	+ 2.4
Brittleness	55.8	55.5	55.7		55.5	57.0	56.2		51.9	51.7	51.8		52.2	52.1	52.2		55.9	52.0	− 3.9
Fuel value	74	74	74		74	71	73		87	85	86		86	87	87		74	86	+12
Weight per cu. ft. (2.5 per cent moisture basis), lb.	31.61		31.61						31.41		31.41						31.61	31.41	− 0.20

[a] Coke basis.

of slate. It has, however, been Clairton's experience that under 20 mesh the disseminated ash, if not excessive, may actually improve the coke structure because of the increased physical strength it gives.

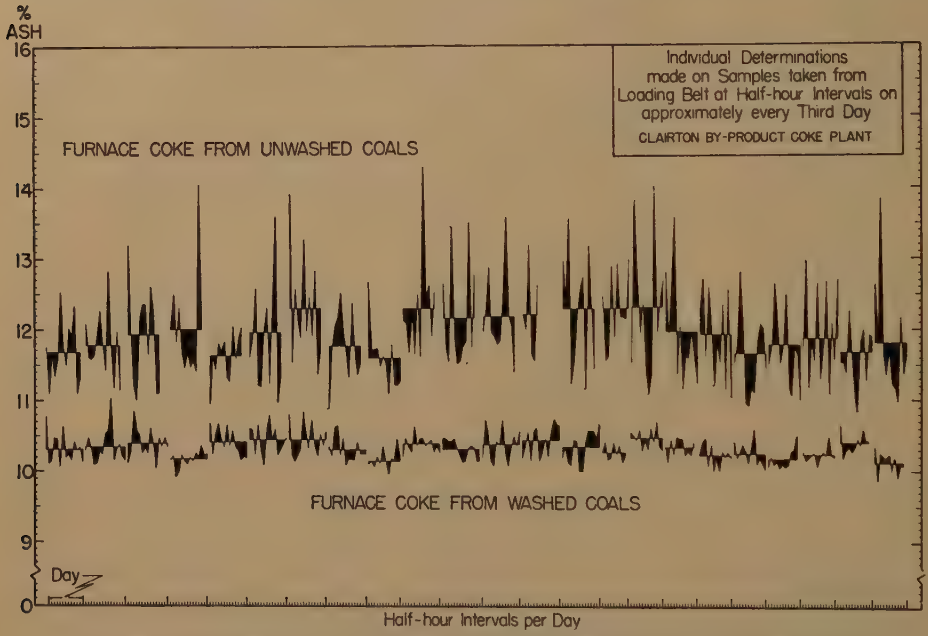


Fig. 1.—Variations in ash of furnace coke from washed and unwashed coals.

The physical tests of furnace coke produced during the four-months period are shown in Table 4. The physical tests relating to coke strength, hardness and brittleness, are conducted according to empirical methods developed at the Clairton by-product coke plant. These methods have proved reliable in determining the relative value of different cokes produced at that plant. Detailed data showing the results for each of the four months are shown in Table 5.

Variation in Sulphur and Ash Content of Furnace Coke

One of the most pronounced benefits obtained from washing Colonial-Ronco coals is the marked improvement in the uniformity of coke ash and sulphur content. Because the large daily production of the Clairton coke plant serves many blast furnaces in the Pittsburgh and Youngstown districts, it was considered inadvisable to use the daily average analyses of coke produced at this plant as the sole criterion of its regularity with respect to blast-furnace results at Carrie. Since the Carrie blast furnaces were not provided with facilities that made it feasible to obtain coke samples at the coke stock bins, arrangements were made to sample at the Clairton coke plant by obtaining samples of coke on the belt conveyors at 30-min. intervals during the day turn. Fig. 1 represents the variations in ash for the four months period for both types of coke. The days indicated represent approximately every third day for the entire four months and each day, in turn, is represented by from 8 to 14 samples. Since the half-hourly production of the Clairton plant on washed coals is equivalent to 200 tons, or a 6 to 8-hr. operation on a single Carrie blast furnace, this frequency of sampling made it possible to study the magnitude of variations of coke during short intervals of time at Clairton.

The study of ash variability may prove of particular value in connection with the treatment of coals in the future, which are expected to be higher in ash and sulphur than the Colonial-Ronco coals used as a basis for this study. This is due to the well known fact that the frequency of slate particles of sufficient size to cause cross fracturing increases at a rate faster than indicated by the rising total average ash content. Likewise, the frequency and severity of short-time irregularities become disproportionately worse where the change in ash content is not caused by change in composition of the coal seam, but rather is due to the inclusion in irregular amounts of extraneous material from bottom and roof of coal seams.

TABLE 6.—*Amount and Variation of Ash and Sulphur in Coke*
PER CENT

Variation for Any Single Day Based on Periodic Daily Tests Using Half-hour Samples for Four-months Period	Furnace Coke Produced from					
	Unwashed Coal			Washed Coal		
	Maximum	Minimum	Range	Maximum	Minimum	Range
Ash	14.05	11.08	2.97	11.04	10.08	0.96
Sulphur	1.08	0.82	0.26	0.97	0.80	0.17

Variation Based on Daily Average Analysis for Every Day of the Four-months Period	Unwashed Coal	Washed Coal
Average ash	11.87	10.36
Average day to day variation in ash	0.40	0.14
Variations exceeding 0.30 per cent ash	50.00	9.50
Variations exceeding 0.50 per cent ash	31.50	0.00
Variations exceeding 1.00 per cent ash	6.70	0.00

Fig. 1 shows a marked and important difference between coke produced from washed and unwashed Colonial-Ronco coals with respect to frequency and magnitude of short-period irregularities in ash content. Similar trends were observed in sulphur content of the respective cokes, but obviously of a smaller character. The reduction in the range of ash variation is largely the result of removal by washing of varying amounts of segregated high-ash fractions in the coal, which otherwise are not leveled out and persist throughout the entire unloading and bunkering system when these coals are carbonized in raw form. Although Fig. 1 indicates a high degree of uniformity in ash content of coke produced from washed coals, it is believed that a further improvement could be expected if the blending-bin capacity of 3000 tons per day were increased to more properly conform with the daily washing capacity of 14,000 tons.

Table 6 shows the amount and variation of ash and sulphur in coke. Based on half-hourly samples, as previously described, the greatest variation in ash content for any one day was 2.97 per cent for coke from unwashed coals, while that for washed coals was only 0.96 per cent and, similarly, the sulphur showed a variation of 0.26 per cent for unwashed coals and only 0.17 per cent for coke from washed coals. A review of the variation based on daily average samples for each day of the four-months test confirms the improved uniformity in coke resulting from washing coals. The coke from unwashed coals shows considerably more severe and frequent variations than the coke from washed coals. For example, day to day variations in excess of 0.30 per cent ash occurred 50 per cent of the time for the former, as opposed to only 9.5 per cent of the time for the latter; and perhaps of greater importance is the fact that no day to day variations in excess of 0.50 per cent ash occurred in coke from washed coals, whereas such occurrences represented 31.5 per cent of the time on coke from unwashed coals.

Comparison of Coke Yields

Four tests were conducted to determine the difference in carbonizing capacity and coke yields when charging washed and unwashed coals. The coal content of 11 larry cars, each of both washed and unwashed coals, was weighed by discharging the coal from the cars into a truck, which later was weighed on the plant scales. Identical volumetric ring settings were maintained throughout the test and the amount of coal pulled back by the leveler bar in the oven was also weighed. The net coal charged per oven, determined in this manner, indicated that the coal-carbonizing capacity of the ovens was reduced approximately 1.0 per cent when washed coals were used. This is shown in Table 7.

TABLE 7.—*Effect of Coal Washing on Oven Carbonizing Capacity*

	Oven Capacities Batteries 19 to 22, Inclusive		Effect of Washing, Per Cent
	Unwashed Coals, Lb.	Washed Coals, Lb.	
Coal as received (3.46 per cent moisture).	41,997.8	41,558.7	−1.05

While several tests were undertaken to determine accurately the total coke yield from both types of coals, in all cases coke over 1 in. being considered furnace coke, this procedure was abandoned because it was impossible to accurately weigh the coal charged as allocated to the different ovens. As a matter of information, this test showed approximately equal total coke yields from washed and unwashed coal, 72.24 per cent and 72.26 per cent, respectively. Since it is well known that a higher total coke yield is obtained with higher-ash coals, it was necessary to disregard these tests and use Clairton's actual experience with washed and unwashed coals as the basis for coke yields. As will be noted in Table 8, a very marked increase was obtained in furnace coke from washed coals, in the order of 2.20 per cent, and the coke dust reduced by 2.90 per cent. However, as was to be expected, the total coke yield for washed coal is less by 0.69 per cent than when using unwashed coals. The increase in yield of furnace coke for washed coal more than offsets the decreased coal-carbonizing capacity resulting from the use of higher-moisture washed coals, and a net increase of approximately 3.0 per cent is obtained per day per battery in the production of furnace coke.

TABLE 8.—*Effect of Coal Washing on Coke Yields*

PER CENT

Coke	Yield per Net Ton of Coal Charged, Per Cent		Effect of Washing
	Unwashed Coals	Washed Coals	
Furnace coke (2.5 per cent moisture basis)	65.48	67.68	+2.20
Coke dust, as produced basis...............	8.29	5.39	−2.90
Quencher sump sludge	0.10	0.11	+0.01
Total coke.........	73.87	73.18	−0.

Comparison of By-product Yields

Since the same by-product plant served the batteries that were concurrently used in the production of coke from washed and unwashed coals, it was not possible to segregate the by-products and determine the yields obtained from the two coals. Therefore, the by-product yields were established by recognizing the amount of by-products lost with float coal in washery refuse, such calculated yields checking past experience at Clairton.

Because of the higher moisture content of washed coals, it is obvious that more oven gas is required for underfiring ovens than when using unwashed coals, and to determine the exact difference a special test was undertaken. Gas-metering facilities were installed on various batteries and, based on an eight-months test, it was determined that 9.0 per cent more fuel was required per ton of washed coal carbonized. The B.t.u. per pound of coal carbonized was 1219 for washed coal—an increase of 102 B.t.u. over that required for unwashed coal. This adversely affected the surplus coke gas available and is included in the determination of the by-product yields shown in Table 9.

Table 9.—*Comparison of By-product Yields from Washed and Unwashed Coals*

	Yield per Net Ton of Coal Charged		Effect of Washing
	Unwashed	Washed	
Tar (160,000 B.t.u. basis), gal.	10.460	10.760	+0.300
Gas (500 B.t.u. basis), M cu. ft.:			
Used at ovens	4.584	4.997	+0.413
Surplus	6.981	6.911	−0.070
Total	11.565	11.908	+0.343
Ammonium sulphate (25 per cent NH_3 basis), lb.	25.38	26.10	+0.72
Light oil (total), gal.	3.16	3.25	+0.09

Effect on Sulphur Content of By-product Coke-oven Gas

Of more than passing interest to steel producers is the fact that by-product coke-oven gas produced from washed coals contained approximately 20 per cent less sulphur than gas produced from unwashed coals, the figures being 0.454 and 0.563 lb. H_2S per M cu. ft., respectively. By-product gas produced at the Clairton Works usually contains more sulphur than that found in similar gas produced elsewhere because of the sulphur content of the coals carbonized. As a result, in the production of steels with low sulphur specifications, special efforts are required through the agency of additional fluxes, manganese, and furnace time, when appreciable proportions of by-product coke-oven gas are used in melting. Therefore a reduction in the sulphur content of the gas would permit the attainment of the required specifications for sulphur in somewhat less melting time and with a reduction in fluxes, manganese, and other materials involved. Indirectly, this matter is important to blast-furnace operators, since it would allow a moderate increase in the permissible sulphur content of pig iron if the present steelmaking practices were retained.

Summary of Four-months Test

The four-months test performed at the Clairton by-product plant to determine the effect of washing coking coals indicated that a marked improvement can be expected in ash, sulphur, physical properties and uniformity of the coke so produced. Blast-furnace operators will readily concede that the use of such coke, as opposed to coke from unwashed coals, should help to improve blast-furnace operations, but only carefully controlled blast-furnace tests can indicate the magnitude of the expected improvement.

Blast-furnace Tests

In the determination of the relative effect of each type of coke on blast-furnace performance, two furnaces of identical physical dimensions were selected, each provided with similar stove and gas-washing facilities. For a long period prior to the test both furnaces had operated with coke from washed coals, with similar operating results and characteristics. The furnaces chosen for the test were Carrie No. 1 and Carrie No. 2, at Rankin, Pa., Carnegie-Illinois Steel Corporation. The essential dimensions are given in Table 10.

Table 10.—*Dimensions of Carrie Blast Furnaces Used in Test of Coke from Washed and Unwashed Coals*

Hearth diameter	22 ft. 6 in.
Bosh diameter	25 ft. 11 in.
Stockline diameter	18 ft. 3 in.
Large bell diameter	13 ft. 3 in.
Bosh angle	81° 0′
Inwall batter	1.313 in.: 12 in.
Total height	92 ft. 0 in.
Hearth area	397.6 sq. ft.
Total volume	33,023 cu. ft.

The initial test comprehended a two-months operation using coke from unwashed coals on one furnace while the companion furnace was operated with coke from washed coals. Since it was recognized that every furnace has certain individual characteristics contributing to its operating efficiency, which may change from time to time, and that identical furnace size and design does not necessarily furnish a guarantee of duplicate performance, a second test of two months duration was considered essential for check purposes, during which the type of coke was interchanged between the furnaces. Between the two test periods, one month was allowed for the adjustment and stabilization of operating practice, and some special tests were also made during this interim for the determination of the amount of coke fines inherent in the two different types of coke used. The following illustrates the test program:

Period	No. 1 Furnace	No. 2 Furnace
July and Aug.	Coke from unwashed coals	Coke from washed coals
September	Change-over and special tests	
Oct. and Nov.	Coke from washed coals	Coke from unwashed coals

The immediate common objective with both types of coke was the production of standard basic iron of similar chemical composition and equal uniformity, the aim being to produce iron containing 1.10 per cent silicon and 0.030 per cent sulphur, normal to Carrie's practice. The test was so arranged as to introduce no variable other than the type of coke. Since the sulphur content of the coke made from unwashed coals is higher than that from washed coals, similar sulphur content of iron from the two blast-furnace units could be accomplished only through a change in slag volume or slag composition.

The required magnitude of the change in slag volume could be readily determined in advance with accuracy from the prescribed sulphur specification as well as the expected sulphur content of the slag, whose composition and sulphur-carrying power corresponded to those normally produced at Carrie when using coke from washed coals. However, the average sulphur content of coke was not the only factor affecting the adjustment in slag volume. Prior experience with coke from washed coals having indicated a marked improvement in regularity of coke composition, it was anticipated that coke made from unwashed coals would show greater variation in sulphur as well as ash content; the resultant adverse effect on chemical composition of iron could be offset only by a further increase in slag volume. This additional slag volume could be obtained either by the use of gravel or a greater proportion of siliceous ore than normally employed when using coke from washed coals. Since this ore was similar in physical characteristics to the other ores, no additional variable was introduced by its use. The additional slag volume was, therefore, obtained by a combination of both practices; i.e., using some gravel and more siliceous ores.

The burdens of miscellaneous materials with metallic content, such as open-hearth slag, excess scrap or scale, were substantially the same on both furnaces throughout the test, and blowing rates were held as constant as practicable, being varied only within the limits imposed by the quality of coke.

First Blast-furnace Test Period

During the first test period, July and August, the performance of No. 2 furnace, using coke from washed coals, was markedly superior to the companion furnace, the iron production being greater by 51.6 tons per day, or 6.7 per cent and the coke consumption correspondingly decreased 157 lb., or 8.1 per cent. Similarly, this furnace operated with 22.6 per cent less flux and with 18.4 per cent less slag volume. Throughout the test No. 1 furnace, using coke from unwashed coals, operated with higher blast pressures than the companion furnace and was more difficult to control, requiring more drastic counteraction against furnace swings for restoration of normal operating conditions.

While the average silicon and sulphur produced in the iron from the respective furnaces for the two-months period was closely similar, the furnace operating with coke from washed coals was characterized by greater uniformity of iron analysis, as indicated by fewer offcasts in silicon and sulphur. Principal data for the first test period appear in Table 11.

TABLE 11.—*Comparative Blast-furnace Performance and Practice Data for First Two-months Test Period*

	Furnace Coke from		Effect of Washing	
	Unwashed Coals	Washed Coals		
	No. 1 Furnace	No. 2 Furnace	Amount	Per Cent
Coke analysis (dry basis), per cent:				
Ash	11.89	10.40	−1.49	
Sulphur	0.99	0.89	−0.10	
Fixed carbon	87.81	89.30	+1.49	
Day to day variations exceeding 0.30 per cent ash in coke, per cent	55.7	18.9	−36.8	
Iron production, net tons per day	769.0	820.6	+51.6	+ 6.7
Net coke rate, lb. per ton iron	1,931	1,774	− 157	− 8.1
Flux rate, lb. per ton iron	987	764	− 223	−22.6
Calculated slag volume, lb. per ton iron	1,285	1,048	− 237	−18.4
Air blown, cu. ft. per min. (carbon basis)	53,422	53,522	+ 100	+ 0.2
Blast pressure, lb. per sq. in	19.3	17.8	− 1.5	
Average iron analysis, per cent:				
Silicon	1.12	1.14	+0.02	
Sulphur	0.028	0.028	0.00	
Casts over 1.40 per cent silicon, per cent	8.75	8.04	−0.71	
Casts under 0.85 per cent silicon, per cent	9.09	4.55	−4.54	
Casts over 0.040 per cent sulphur, per cent	6.40	4.90	−1.50	

Second Blast-furnace Test Period

During the second test period, covering October and November, the type of coke used on the two furnaces was reversed, No. 1 operating with coke from washed coals and No. 2 with coke from unwashed coals. No. 1 furnace showed an improved performance over the companion furnace, producing 74.7 tons more iron per day, or 9.4 per cent, and using 139 lb. less coke per ton of iron, or 7.4 per cent. This furnace required 17.5 per cent less flux and operated with a 12 per cent lower slag volume. During this period the blowing rate on No. 1 furnace was increased approximately 2.7 per cent over the companion furnace, a blowing rate permitted by the improved quality of the coke.

The furnace operating with coke from washed coals also demonstrated a marked superiority with respect to regularity of iron analysis, since only 1.00 per cent of the casts for this period exceeded 1.40 per cent silicon as opposed to 9.19 per cent for the companion furnace. Similarly, only 2.70 per cent casts exceeded 0.040 per cent sulphur compared with 5.78 per cent for the other furnace. Table 12 shows the results obtained during the second test period.

TABLE 12.—*Comparative Blast-furnace Performance and Practice Data for Second Two-months Test Period*

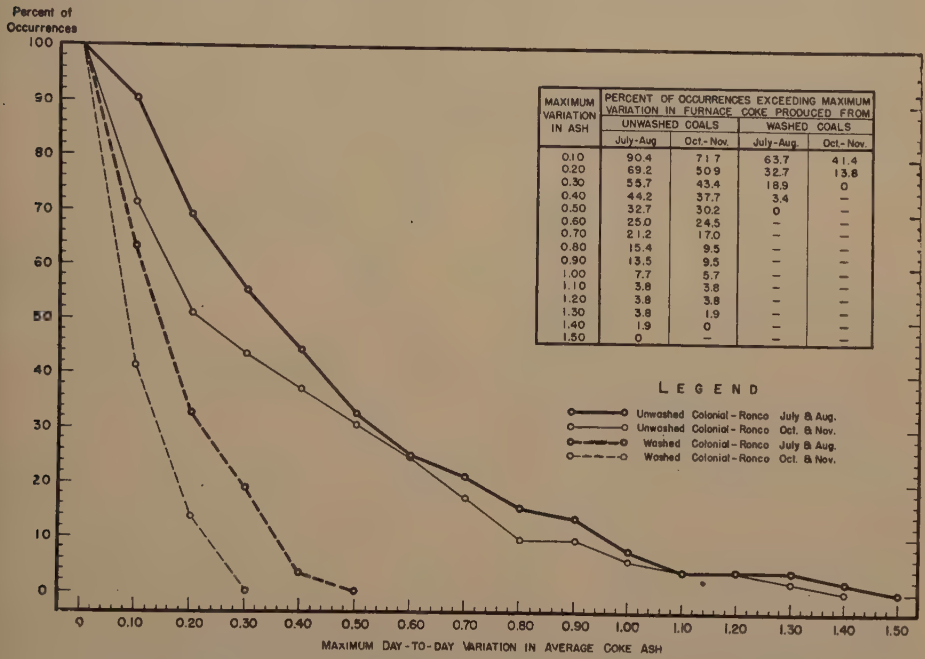
	Furnace Coke from		Effect of Washing	
	Unwashed Coals	Washed Coals		
	No. 2 Furnace	No. 1 Furnace	Amount	Per Cent
Coke analysis (dry basis), per cent				
Ash	11.85	10.33	−1.52	
Sulphur	0.98	0.88	−0.10	
Fixed carbon	87.85	89.37	+1.52	
Day to day variations exceeding 0.30 per cent ash in coke, per cent	43.4	0	−43.4	
Iron production, net tons per day	793.5	868.2	+74.7	+ 9.4
Net coke rate, lb. per ton iron	1,887	1,748	− 139	− 7.4
Flux rate, lb. per ton iron	1,003	827	− 176	−17.5
Calculated slag volume, lb. per ton iron	1,287	1,132	− 155	−12.0
Air blown, cu. ft. per min. (carbon basis)	53,301	54,721	+1420	+ 2.7
Blast pressure, lb. per sq. in.	19.4	19.2	− 0.2	
Average iron analysis, per cent:				
Silicon	1.14	1.05	−0.09	
Sulphur	0.029	0.028	−0.001	
Casts over 1.40 per cent silicon, per cent	9.19	1.00	−8.19	
Casts under 0.85 per cent silicon, per cent	8.16	7.72	−0.44	
Casts over 0.040 per cent sulphur, per cent	5.78	2.70	−3.08	

Comparison of Furnace Performance on Same Grade of Coke during the Two Periods

Of considerable interest is the effect of uniformity of coke quality on blast-furnace performance. This is illustrated by a comparison of the results obtained on the same grade of coke for the two test periods, as shown in Table 13. Pig-iron production using coke from unwashed coals increased 3.2 per cent during the second period, compared with the first period when using similar coke, whereas a 5.8 per cent improvement was obtained on coke produced from washed coals. The increase in wind rate of 2.2 per cent permitted by coke from washed coals when comparing the two periods was unaccompanied by a similar increase when comparing the two periods using coke from unwashed coals. The coke consumption decreased for both grades of coke used during the second period, but the magnitude of the change is not sufficient to account for the difference in production performance, particularly in view of the change in other related factors. The distribution of iron analysis also improved during the second period, the magnitude of the improvement being greater on coke from washed coals.

There are three distinct differences between the periods: (1) the use of equivalent amounts of sinter in the second period; (2) the lower atmospheric moisture content in the second period; (3) a major improvement in the regularity of coke produced from washed coals and a more moderate improvement in coke produced from unwashed coals, as subsequently illustrated. The use of 4.6 per cent sinter in the burden of both furnaces during the second period could account for only a minor part of the relative improvement shown by both furnaces over the first period. While some operators may believe that some of the improvement is due to the lower atmospheric humidity during the second period, this does not explain the relatively greater improvement obtained during that period with coke from washed coals as opposed to coke from unwashed coals.

Since the differences in sinter charged and atmospheric humidity were common to the two grades of coke, it follows that by far the greater part of the general improvement in the second period must be credited to the greater uniformity of ash in coke produced from washed coals and the relatively moderate increase in uniformity of ash in the companion coke. A statistical study of coke-ash variability for the two periods, based on differences in day to day average ash content of the respective cokes, reveals the pronounced improvement in ash uniformity of coke from washed coals over coke from unwashed coals and, of equal importance, shows improvement in uniformity of both grades of coke during the second test period. The data and graph shown in Fig. 2 demonstrate that the degree of improvement was far greater for coke from washed coals, as evidenced by the difference in number of occurrences exceeding a maximum day to day variation of 0.30 per cent coke ash. Occurrences exceeding this severity of variation decreased 12.3 per cent for coke from unwashed coals when comparing the second period with the first test period, whereas the improvement, on a similar basis, was 18.9 per cent for the coke from washed



MAXIMUM VARIATION IN ASH	PERCENT OF OCCURRENCES EXCEEDING MAXIMUM VARIATION IN FURNACE COKE PRODUCED FROM			
	UNWASHED COALS		WASHED COALS	
	July-Aug	Oct.-Nov.	July-Aug.	Oct.-Nov.
0.10	90.4	71.7	63.7	41.4
0.20	69.2	50.9	32.7	13.8
0.30	55.7	43.4	18.9	0
0.40	44.2	37.7	3.4	–
0.50	32.7	30.2	0	–
0.60	25.0	24.5	–	–
0.70	21.2	17.0	–	–
0.80	15.4	9.5	–	–
0.90	13.5	9.5	–	–
1.00	7.7	5.7	–	–
1.10	3.8	3.8	–	–
1.20	3.8	3.8	–	–
1.30	3.8	1.9	–	–
1.40	1.9	0	–	–
1.50	0	–	–	–

FIG. 2.—COMPARATIVE DAY TO DAY VARIATIONS IN AVERAGE DAILY ASH OF FURNACE COKE FROM WASHED AND UNWASHED COALS, CLAIRTON BY-PRODUCT COKE PLANT.

coals; in fact, no such variations occurred in coke produced from washed coals.

The reduction in coke-ash variability logically not only explains the improved operating performance on both grades of coke during the October and November period but may be interpreted as a primary factor in the greater magnitude of improvement in furnace performance on coke from washed coals in that period.

TABLE 13.—*Comparison of Blast-furnace Performances*

	Coke from Unwashed Coals				Coke from Washed Coals			
	July, August	October, November	Difference		July, August	October, November	Difference	
			Amount	Per Cent			Amount	Per Cent
Coke analysis (dry basis), per cent:								
Ash	11.89	11.85	−0.04		10.40	10.33	−0.07	
Sulphur	0.99	0.98	−0.01		0.89	0.88	−0.01	
Fixed carbon	87.81	87.85	+0.04		89.30	89.37	+0.07	
Moisture	3.01	2.90	−0.11		2.90	2.86	−0.04	
Day to day variations exceeding 0.30 per cent ash in coke, per cent	55.7	43.4	−12.3		18.9	0	−18.9	
Sinter in ore and sinter burden, per cent	0	4.6	+4.6		0	4.6	+4.6	
Air blown, cu. ft. per min. (carbon basis)	53,422	53,301	−121	−0.2	53,522	54,721	+1199	+2.2
Blast temperature, deg. F	1,085	1,032	−53		1,116	1,080	−36	
Top temperature, deg. F	359	379	+20		414	388	−26	
Blast pressure, lb. per sq. in.	19.3	19.4	+0.1		17.8	19.2	+1.4	
Moisture in air, grains per cu. ft.	6.00	2.85	−3.15		6.00	2.85	−3.15	
Iron production, net tons per day	769.0	793.5	+24.5	+3.2	820.6	868.2	+47.6	+5.8
Net coke rate, lb. per ton iron	1,931	1,887	−44	−2.3	1,774	1,748	−26	−1.5
Flux rate, lb. per ton iron	987	1,003	+16	+1.6	764	827	+63	+8.2
Flue dust and sludge produced, lb. per ton	139	142	+3		158	123	−35	
Calculated slag volume, lb. per ton iron	1,285	1,287	+2	+0.2	1,048	1,132	+84	+8.0
Ratio bases to acids in slag	1.044	1.048	+0.004		1.024	1.035	+0.011	
Sulphur in slag, per cent	1.79	1.86	+0.07		1.81	1.83	+0.02	
Average iron analysis, per cent:								
Silicon	1.12	1.14	+0.02		1.14	1.05	−0.09	
Sulphur	0.028	0.029	+0.001		0.028	0.028	0	
Casts over 1.40 per cent silicon, per cent	8.75	9.19	+0.44		8.04	1.00	−7.04	
Casts under 0.85 per cent silicon, per cent	9.09	8.16	−0.93		4.55	7.72	+3.17	
Casts over 0.040 per cent sulphur, per cent	6.40	5.78	−0.62		4.90	2.70	−2.20	

Determination of Coke Screenings

The coke-screening facilities at Carrie Nos. 1 and 2 blast furnaces are so arranged that the fines removed from the furnace coke discharge on a common conveyor for transfer to a common collecting bin. Since this arrangement did not permit separation of coke screenings produced at each unit during the test, both furnaces were operated on coke from washed coal during part of the interim period of September, and the coke screenings were determined. The total screenings recovered from both grades of coke at the blast furnaces during the four-months test period were known and the figure ascertained by the special test was used as a basis for determining the screenings produced from coke from washed coals. The screenings recovered from coke from unwashed coals were obtained by difference. In all cases coke dust under ¾ in. was considered screenings. The essential data are shown in Table 14.

TABLE 14.—*Blast-furnace Coke Screenings Recovered at Carrie Blast Furnaces for Four-months Test Period*

	Furnace Coke from		Effect of Washing
	Unwashed Coals	Washed Coals	
Coke screenings:			
Per ton of coke, lb. .	59.3	42.5	−16.8
Per ton of iron, lb. .	57.0	38.5	−18.5
Coke required per ton of iron:			Per Cent
Gross furnace coke.	1,966	1,799	− 8.5
Net furnace coke. . .	1,909	1,760	− 7.8

Summary of Results at Blast Furnaces

Discussion of results obtained in the two separate periods confirms the need for duplicate checks such as were undertaken, and is indicative of differences in magnitude of benefits that may be obtained over an extended period. The periods involved reasonably approximate the average conditions that may obtain throughout the year, therefore the composite of the four-months test, as shown in Table 15, represents a sound basis for evaluating the advantages of coke produced from washed coals. The following advantages were obtained by the use of such coke.

1. Daily iron production was increased 63.3 tons, or equivalent to 8.1 per cent.

2. Net furnace coke consumption was decreased 149 lb., or equal to 7.8 per cent.

3. Flux consumption was decreased 198 lb., or equal to 19.9 per cent, and slag volume was reduced 15.2 per cent.

4. A pronounced improvement was obtained in the regularity of pig-iron analysis.

In addition, an item of considerable importance is the fact that the increased yield of furnace coke of 3.0 per cent at the Clairton coke plant, combined with the subsequent reduction in coke requirements per ton of iron, means that less mined coal is required per ton of iron under such conditions than when using coke from unwashed coals. In the former case 1.375 net tons of mined coal were required and in the latter 1.501, or a difference equal to 8.4 per cent less mined coal needed when using washed coals.

TABLE 15.—*Comparative Blast-furnace Performance and Practice Data for Composite Four-months Test Period*

	Furnace Coke from		Effect of Washing	
	Unwashed Coals	Washed Coals	Amount	Per Cent
Coke analysis (dry basis), per cent:				
Ash	11.87	10.36	−1.51	
Sulphur	0.98	0.88	−0.10	
Fixed carbon	87.83	89.34	+1.51	
Day to day variations exceeding 0.30 per cent ash in coke, per cent	50.0	9.5	−40.5	
Iron production, net tons per day	781.1	844.4	+63.3	+8.1
Net coke rate, lb. per ton iron	1,909	1,760	−149	−7.8
Flux rate, lb. per ton iron	995	797	−198	−19.9
Calculated slag volume, lb. per ton iron	1,286	1,091	−195	−15.2
Air blown, cu. ft. per min. (carbon basis)	53,362	54,129	+767	+1.4
Blast pressure, lb. per sq. in	19.3	18.6	−0.7	
Average iron analysis, per cent:				
Silicon	1.13	1.09	−0.04	
Sulphur	0.029	0.028	−0.001	
Casts over 1.40 per cent silicon, per cent	8.97	4.45	−4.52	
Casts under 0.85 per cent silicon, per cent	8.63	6.17	−2.46	
Casts over 0.040 per cent sulphur, per cent	6.09	3.77	−2.32	

TABLE 16.—*Comparative Blast-furnace Performance and Practice Data at Carrie Furnaces*

	First Test Period, July and August				Second Test Period, October and November				Composite Test Period, July, August, October and November			
	Coke from		Difference		Coke from		Difference		Coke from		Difference	
	Unwashed Coals	Washed Coals	Amount	Per Cent	Unwashed Coals	Washed Coals	Amount	Per Cent	Unwashed Coals	Washed Coals	Amount	Per Cent
Coke analysis (dry basis), per cent:												
Ash	11.89	10.40	−1.49		11.85	10.33	−1.52		11.87	10.36	−1.51	
Sulphur	0.99	0.89	−0.10		0.98	0.88	−0.10		0.98	0.88	−0.10	
Fixed carbon	87.81	89.30	+1.49		87.85	89.37	+1.52		87.83	89.34	+1.51	
Moisture	3.01	2.90	−0.11		2.90	2.86	−0.04		2.95	2.88	−0.07	
Day to day variations exceeding 0.30 ash, per cent	55.7	18.9	−36.8		43.4	0	−43.4		50.0	9.5	−40.5	
Burden, lb. per ton iron:												
Group 3 ore	1,800	2,004	+204		2,744	2,820	+76		2,276	2,424	+148	
Group 7 ore	658	490	−168		507	398	−109		582	443	−139	
Beaver ore	959	972	+13		19	18	−1		486	481	−5	
Sintered flue dust	0	0	0		158	157	−1		80	80	0	
Pit cinder	26	26	0		39	37	−2		33	31	−2	
Roll scale	50	51	+1		57	53	−4		53	52	−1	
Open-hearth slag	384	373	−11		349	405	+56		366	390	+24	
Excess scrap	5	2	−3		6	1	−5		6	5	−1	
Gravel	43	4	−39		74	5	−69		58	4	−54	
Limestone	783	606	−177		749	618	−131		766	612	−154	
Dolomite	204	158	−46		254	209	−45		229	185	−44	
Production:												
Iron, net tons per day	769.0	820.6	+51.6	+6.7	793.5	868.2	+74.7	+9.4	781.1	844.4	+63.3	+8.1
Flue dust and sludge, lb. per ton iron	139	158	+19		142	123	−19		141	140	−1	
Consumption:												
Net coke, lb. per ton iron	1,931	1,774	−157	−8.1	1,887	1,748	−139	−7.4	1,909	1,760	−149	−7.8
Gross coke, lb. per ton iron	1,989	1,812	−177	−8.9	1,945	1,787	−158	−8.1	1,966	1,799	−167	−8.5
Flux, lb. per ton iron	987	764	−223	−22.6	1,003	827	−176	−17.5	995	797	−198	−19.9
Iron analysis, per cent:												
Silicon	1.12	1.14	+0.02		1.14	1.05	−0.09		1.13	1.09	−0.04	
Sulphur	0.028	0.028	0		0.029	0.028	−0.001		0.029	0.028	−0.001	
Casts over 1.40 per cent silicon	8.75	8.04	−0.71		9.19	1.00	−8.19		8.97	4.45	−4.52	
Casts under 0.85 per cent silicon	9.09	4.55	−4.54		8.16	7.72	−0.44		8.63	6.17	−2.46	
Casts over 0.040 per cent sulphur	6.40	4.90	−1.50		5.78	2.70	−3.08		6.09	3.77	−2.32	
Slag:												
Analysis, per cent:												
Silica	34.80	34.34			34.54	33.95			34.67	34.15		
Alumina	12.87	13.71			12.93	13.79			12.90	13.76		
Lime	43.91	43.21			43.51	42.73			43.71	42.93		
Magnesia	5.87	5.98			6.22	6.70			6.03	6.33		
Sulphur	1.79	1.81	+0.02		1.86	1.83	−0.03		1.83	1.82	−0.01	
Ratio bases to acids	1.044	1.024	−0.020		1.048	1.035	−0.013		1.046	1.028	−0.018	
Calculated volume, lb. per ton iron	1,285	1,048	−237	−18.4	1,287	1,132	−155	−12.0	1,286	1,091	−195	−15.2
Practice:												
Air blown, cu. ft. per min. (carbon basis)	53,422	53,522	+100	+0.2	53,301	54,721	+1,420	+2.7	53,362	54,129	+767	+1.4
Blast pressure, lb. per sq. in.	19.3	17.8	−1.5		19.4	19.2	−0.2		19.3	18.6	−0.7	
Blast temperature, deg. F.	1,085	1,116	+31		1,032	1,080	+48		1,058	1,098	+40	
Top temperature, deg. F.	359	414	+55		379	388	+9		368	402	+34	
Moisture in air, grains per cu. ft.	6.00	6.00	0		2.85	2.85	0		4.43	4.43	0	

The increase in production of iron and the reduction in coke rate shown in Table 15 can be attributed wholly to the improvement in quality of coke from washed coals.

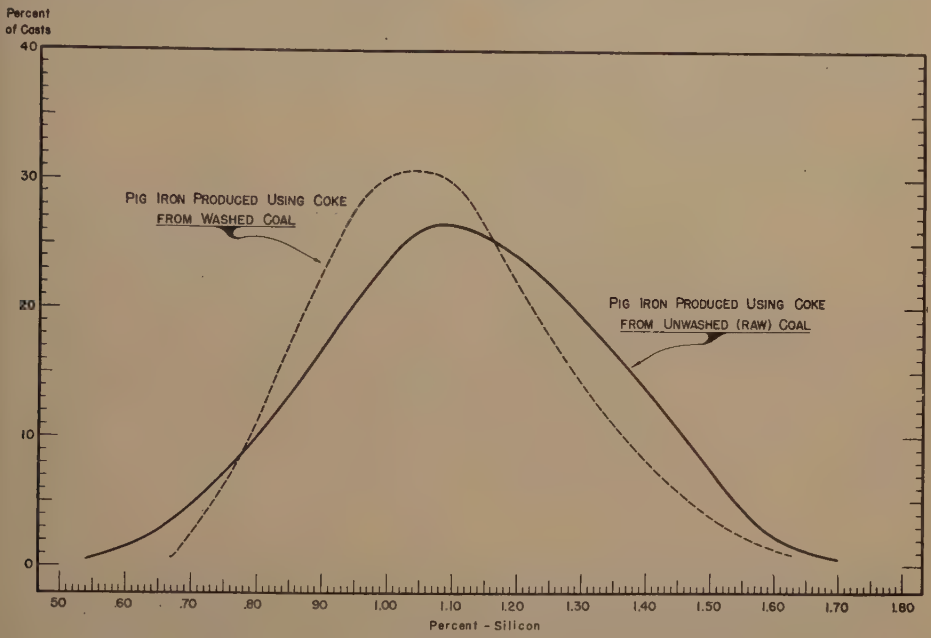


FIG. 3.—COMPARATIVE DISTRIBUTION OF SILICON IN PIG IRON PRODUCED USING FURNACE COKE FROM WASHED AND UNWASHED COALS, CARRIE FURNACES.

The decrease in amount and variability of ash and sulphur in coke and the improvement in physical coke quality, all of which permitted a reduction in flux consumption, slag volume and basicity, and marginal burden reserve, may be cited as major factors in decreasing the coke rate. The precise extent to which each of these factors contributed to the over-all result cannot be readily determined. No marked difference was noted in amount of flue dust produced, although we believe a more extended test would be required to determine accurately the true effect on that of several cokes tested. Complete data for the entire blast-furnace test period, including details relating to slag composition and burdens, appear in Table 16.

The effect of coke from washed coals on furnace operation was reflected not only in improved production of iron and consumption of coke, but was also apparent in the improvement of the regularity of iron analysis, as shown by Figs. 3 and 4, representing distribution of silicon and sulphur analyses of the respective pig irons for the entire period. The fact that the average iron analysis over the entire test period was almost the same for all grades of coke, reflects creditably on the achievement of the blast-furnace operator charged with the responsibility of complying with the severely prescribed conditions under which the test was conducted.

Lower tuyere losses characterized operation with coke from washed coals and, further, the use of coke from washed coals permitted operation with a degree of freedom from irregularity impossible of attainment with coke from unwashed coals. It is difficult to convey adequately the importance of this factor by means of tabulations and graphs; nevertheless, the blast-furnace operator faced with various production problems, many of which suddenly arise owing to unpredictable changes in raw materials, will regard this phase of the test as of the greatest significance.

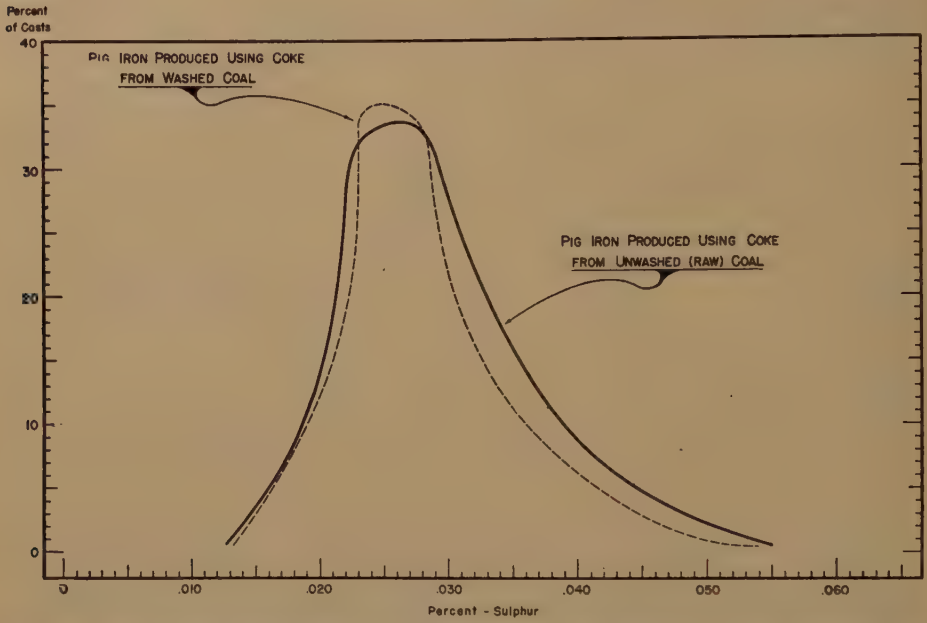


FIG. 4.—COMPARATIVE DISTRIBUTION OF SULPHUR IN PIG IRON PRODUCED USING FURNACE COKE FROM WASHED AND UNWASHED COALS, CARRIE FURNACES.

One word of caution appears to be in order. The benefits cited for washing the specific Pittsburgh seam coals under study do not apply to all coals or all conditions. While the test results closely agree with the blast-furnace operator's rule of thumb—that a decrease of 1.0 per cent in coke ash, above a certain minimum, permits an increase of 5.0 per cent in furnace production—the figures reported should not be applied without qualification to determine the desirability of washing metallurgical coals. The type of mining, the uniformity and grade of coal mined, the physical properties of coke, the degree of uniformity as well as amount of sulphur and ash, will all have a bearing on the magnitude of benefits to be derived from washing coals for furnace coke.

CONCLUSION

The test was undertaken to determine on a practical plant scale the advantages accruing to blast-furnace operation by using coke produced from washed coals as compared with coke produced from unwashed coals. Since it was expected that the magnitude of these benefits would be pronounced, any unpredictable omissions or minor discrepancies that did result can well be disregarded in judging the value of washing the coals in question. In retrospect, we can recognize that detailed information might have been advantageously developed on some phases of this investigation to make this study a more truly technological effort, but we believe that these minor departures can have no bearing on the conclusions that follow from this investigation.

It is our belief that under the conditions outlined, the expected deterioration in quality of coking coals from the Pittsburgh seam accelerated by extension of mine mechanization axiomatically involves coal washing, and that in some cases the potential advantages of mine mechanization cannot be fully realized without supplementing such practice with coal-washing facilities.

ACKNOWLEDGMENTS

The conduct of the test at the Clairton coke plant was under the direct supervision of Mr. D. P. Finney, Division Superintendent Coke Plant, Clairton Works, and the blast-furnace investigation at Carrie furnaces was the responsibility of Mr. C. P. Clingerman, Superintendent Blast Furnaces, Carrie Furnaces, Carnegie-Illinois Steel Corporation. To them is due the credit for a truly fine accomplishment, since, thanks to their efforts, data permitting an unqualified conclusion on a problem of present-day importance was obtained.

We similarly wish to recognize the assistance of Mr. F. M. Becker, United States Steel Corporation of Delaware, who cooperated in many parts of the coke-plant test and was also responsible for the development of much of the data that appear in this report.

Appreciation is also expressed to various blast-furnace operators and other technologists of the subsidiary companies of United States Steel Corporation who assisted in various phases of this test.

The Operation of a Froth Flotation Pilot Plant on Washery-water Solids

By C. D. Rubert,* Member, and W. J. Parton,† Junior Member A.I.M.E.

(New York Meeting, February 1944)

Large quantities of fine anthracite with associated impurities are discharged with the waste water from the coal-cleaning plants in the Pennsylvania anthracite region. Furthermore, for many years this fine material has accumulated and formed silt banks. The difficulty of removing the impurities from the fine material and the lack of adequate markets are responsible for the waste of this material.

In view of a larger demand for fine coal, mainly for pulverized fuel, the Lehigh Navigation Coal Co. Inc. decided to investigate the froth flotation process for recovery of fine coal from these waste solids.

A test plant was constructed and operated for six months during 1942 in order to determine the practicability of recovering coal from the waste water of a cleaning plant (breaker) by froth flotation, and to obtain operating data for designing a commercial plant if the results were favorable.

Preliminary Investigation

Before the pilot plant was constructed information was obtained by field and small-scale laboratory tests to serve as a guide in the design and operation of the pilot plant.

Size and ash analyses of a representative sample of the material discharged in the wash water from a cleaning plant are given in Table 1.

Table 1.—*Representative Sample of Material Discharged in Wash Water of Cleaning Plant*

Mesh	Screen Analysis Weight, Per Cent	Ash Content, Per Cent
+14	1.0	12.5
+20	2.0	
+28	3.8	14.4
+35	6.1	
+48	7.2	18.0
+65	8.0	
+100	10.6	22.9
+150	8.8	
+200	8.8	
−200	43.7	43.26
	100.0 Calculated	29.9
	Head sample	31.49

The refuse content increases progressively as the size of the particles decreases. The larger particles (+35 m.) are low in ash content and could be marketed without cleaning.

Since these solids are discharged from the breakers with 90 to 95 per cent water by weight, the first step in the recovery of the coal was that of reducing the quantity of water so that water in excess will not have to be handled. Furthermore, it was found that the removal of most of the minus 200 mesh high-ash material was desirable to permit the most efficient flotation of the coal. Classification tests were run and showed that a hydroseparator overflow rate of 12 ft. per hour* was sufficient to eliminate much of the minus 200-mesh material while allowing the coarser material to settle and be thickened.

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* See discussion, page 190.

Laboratory flotation tests were conducted to determine the most efficient combination of reagents, quantity and points of addition. Pine oil or an alcohol frother with a light fuel oil as the collector were found to be the most efficient reagents. Introduction by numerous stages was found best. Most of the coal was recovered during the first few minutes of contact time whereas recovery of the remaining coal required considerably more flotation time and greater consumption of reagents.

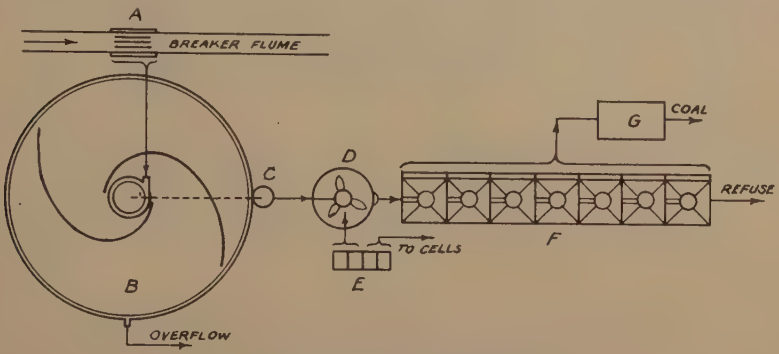

FIG. 1.—LAYOUT OF PILOT PLANT.

Dewatering of the coal froth or concentrates was investigated by natural drainage and filtration. Natural drainage was slow and mechanical dewatering of some form appeared necessary.

LAYOUT OF PILOT PLANT

Fig. 1 shows the pilot-plant equipment and layout. The feed to the pilot plant was tapped out of the main discharge flume by inserting three slicers *A* through slots in the bottom of the trough. About 10 per cent of the total flow was split for testing in the plant. The water flowed to a 22-ft. diameter hydroseparator *B*, which overflowed most of the minus 200-mesh particles and raked the settled solids to the center of the tank. A diaphragm pump *C* elevated the thickened, deslimed solids from the tank and discharged the thickened material to a 6-ft. conditioner tank *D*.

Reagents from reagent feeder *E* were mixed with the pulp in the conditioner and the conditioned pulp then flowed to flotation cells *F*. The clean coal was removed from the cells and flowed directly to a dewatering filter or screen *G*. The refuse flowed directly to waste.

OPERATING DATA

Hydroseparator

Recovery of the solids from the washery water and concurrent elimination of the finest particles of highest ash content with the overflow was accomplished in the 22-ft. hydroseparator. Removal of

TABLE 2.—*Hydroseparator Operation*

Data	Hydro Feed	Underflow	Overflow
Cubic feet per hour	5140	250	4890
Percentage of solids	4.04	47.6	2.05
Tons solids per hr.	7.55	4.45	3.10
Percentage of recovery	100.00	58.5	41.5
Percentage of ash	25.65	21.92	37.60
SCREEN ANALYSIS, PER CENT WT.			
+14 mesh	0.2	0.7	0
+20	1.2	2.2	0
+28	2.2	3.6	0
+35	4.8	6.6	0.1
+48	6.8	9.4	0.1
+100	19.5	27.9	0.4
+200	17.5	22.8	3.2
−200	47.8	26.8	96.2
	100.0	100.0	100.0
Tons −200-mesh solids	4.17	1.19	2.98
Recovery −200-mesh solids		28.5	71.5
Classifying rate, ft. per hr.			13.5

the colloidal clays with a minimum loss of plus 200-mesh coal was attempted. In Table 2 are data showing operating performance after changes were made at the feed well to minimize turbulençe. These data show that only a small percentage of plus 200-mesh coal is lost in the classifier overflow whereas 71.5 per cent of the minus 200-mesh material is eliminated.

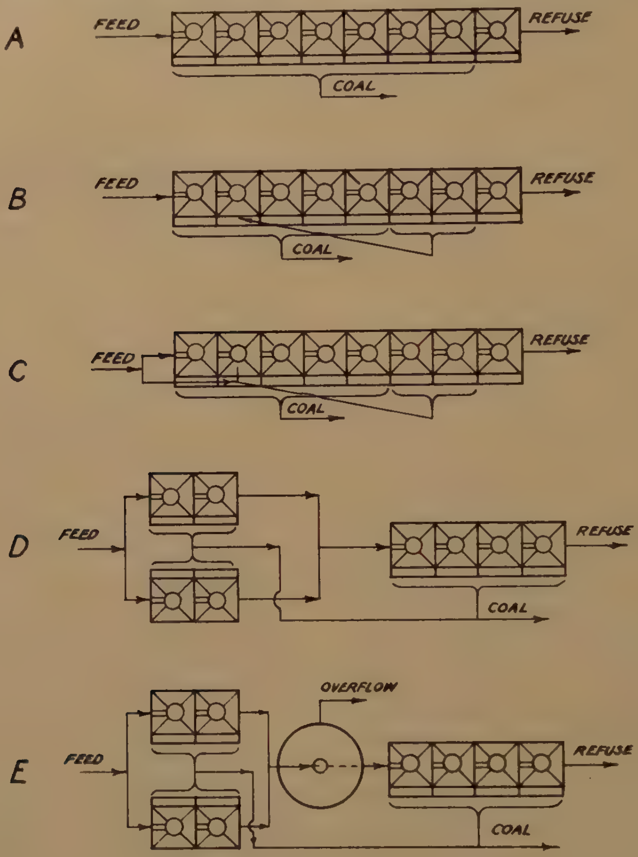

FIG. 2.—FLOTATION CIRCUITS TESTED.

Flotation Cells

A Denver Sub-A (Fahrenwald) flotation machine consisting of eight cells, each of 24 cu. ft. cell volume, was used at the pilot plant. Flotation circuits that were tested in order to determine most efficient use of the cells are shown in Fig. 2.

Circuit *A* was tested first, since it consisted of the simplest arrangement. The feed enters No. 1 cell and is re-treated in each successive cell while coal froth is removed from each cell. The flotation characteristics of anthracite can be explained by using the results of this circuit. Table 3 gives data for one day of operation and shows that 85 per cent of the feed was recovered as a coal froth at 13.7 per cent ash. The refuse contained little coal and was discharged in a diluted state, owing to the removal of the large quantity of coal.

The flotation time or period of contact in the cells was 15.6 min. Reagents were added to the conditioning tank ahead of the cells and to individual cells as needed to maintain flotation. Approximately one third of the total quantity of reagents was added to the conditioner and the remaining two thirds was distributed to the cells.

TABLE 3.—*Operating Data for Froth Flotation Pilot Plant, Circuit A*

Product	Slurry, Tons per Hr.	Solids in Slurry, Per Cent	Solids, Tons per Hr.	Ash, Per Cent Wt.	Distribution Solids, Per Cent Wt.
Hydroseparator feed......................	135	5.21	7.03	28.50	100.0
Hydroseparator overflow..................	123	2.54	3.12	39.07	44.4
Cell feed-hydro underflow[a]...............	22.2	17.61	3.91	23.31	55.6
Cell clean coal...........................	8.5	39.20	3.32	13.70	47.2
Cell refuse................................	13.7	4.30	0.59	77.41	8.4

Consumption of Reagents				General Operating Data
Oil	Lb. per Hr.	Lb. per Ton Coal	Cost, Cents per Ton	
Pine......	2.28	0.69	5.71	Operation, 7½ hr. Number cells operated, 7; flotation time, 15.6. Power,[b] kw-hr. 160; kw. per ton, 6.42 Coal recovery, per cent feed, 85.0 Recovery of combustible, per cent, 95.6 Efficiency (sink and float), per cent, 94.7
Fuel......	5.86	1.76	1.96	

[a] Hydro underflow with dilution water.
[b] No filtering equipment.

TABLE 4.—*Size Analysis and Ash Distribution on Products, Circuit A*

Screen Analysis, Mesh	Hydro Feed		Hydro Overflow		Hydro Underflow (Cell Feed)		Clean Coal		Cell Refuse	
	Per Cent Wt.	Per Cent Ash	Per Cent Wt.	Per Cent Ash	Per Cent Wt.	Per Cent Ash	Per Cent Wt.	Per Cent Ash	Per Cent Wt.	Per Cent Ash
+10	0.3	13.2	0.0	28.51	0.4	13.23	0.4	8.97	0.5	17.15
+20	2.7	11.1	0.3		2.7	11.10	3.0	8.43	1.5	23.89
+35	8.9	10.4	0.8		11.3	10.31	12.8	8.41	3.5	34.16
+65	14.9	13.4	1.6		23.5	13.18	26.1	10.37	8.9	55.12
+150	19.8	21.6	3.3		33.2	21.16	33.9	13.48	25.0	79.14
+200	6.0	28.1	2.8		8.4	27.94	8.1	15.94	10.2	83.16
−200	47.4	40.7	91.2	40.56	20.5	41.15	15.7	21.74	50.4	85.67
Total......	100.0	28.5	100.0	39.07	100.0	23.31	100.0	13.70	100.0	77.41

The size analyses and ash distribution of the various plant products are given in Table 4. The hydroseparator overflow reduced the quantity of high-ash minus 200-mesh material from 47.4 per cent in the feed to 20.5 per cent by weight in the underflow, with a corresponding reduction in ash content from 28.5 to 23.31 per cent. The analysis of the cleaned coal shows that the ash increases as the size decreases. The refuse analysis shows that some coal is lost in the larger sizes.

In order to determine the performance of individual cells and thus learn the economical limit of re-treating the pulp, the froth production of coal from each cell was measured. Table 5 gives these data and shows that maximum production is obtained from the first cell and drops off rapidly thereafter. The realization per cell can then be calculated by deducting the operating costs (including reagents, power and repairs) from the value of the coal produced from each cell. Under the conditions of this test, production from No. 7 cell was insufficient to cover operating costs whereas production from Nos. 5 and 6 cells barely covered expenses. The optimum flotation time was 12.5 minutes.

Because of the large removal of coal from the pulp in the first few cells, the percentage of solids in the pulp for subsequent re-treatment is considerably reduced. Table 5 shows this drop as well as the progressive increase in ash of the recovered coal toward the refuse end of the cells.

TABLE 5.—*Individual Cell Production of Coal, Circuit A*

Data	Cell No.							Total
	1	2	3	4	5	6	7	
Coal, tons per hr.	2.29	0.54	0.14	0.08	0.04	0.05	0.02	3.16
Cell output, per cent total production	72.47	17.09	4.43	2.53	1.27	1.58	0.63	100.0
Ash, per cent wt.	11.49	13.84	16.18	22.26	30.60	30.22	45.45	13.14
Solids in pulp, per cent wt. (feed)[a]	16.0	7.8	5.2	4.5	4.2	4.0	3.7	3.7 (refuse)

[a] These data show the solids by weight that remained in the water-solids mixture (pulp) as it flowed from the feed end of the flotation machine successively through each cell to the refuse discharge.

To improve the efficiency of recovering the coal and reduce the quantity of reagents and the contact time, other circuits shown in Fig. 2 were tested. Circuit *B* returned the higher ash concentrate from the last few cells to No. 2 cell. Returning part of the froth served several purposes; i.e., recleaning this high-ash material to improve the quality of the concentrate, reintroduction of concentrated froth to serve for seeding purposes, and more efficient use of reagents by utilizing the excess oil required to float the large particles in the last few cells. Recirculation of part of the froth also acted as a flywheel to maintain more uniform flotation conditions.

Circuit *C* was tested to determine the effect of maintaining higher pulp densities by introducing half of the feed pulp into the second cell. Recoveries were somewhat lower for this circuit, as production from No. 2 cell was depressed by mixing of unfloated material from the first cell with the fresh feed.

Since highest production is made during the first few minutes of flotation time, circuits *D* and *E* were used in order to have two independent rougher circuits that would recover most of the coal and then discharge the diluted refuse pulp, still containing considerable coal, into a bank of scavenger cells for final removal of coal. By re-treating this dilute, partially depleted pulp from the primary or rougher cells, attention could be given to increasing the efficiency of recovering the more difficult coal to float. Opening the recirculation gates in the cells themselves, maintaining a layer of froth on the pulp surface to eliminate "boiling" and to serve for seeding purposes, and returning the froth from the last few cells to the first cell in the scavenger bank or rougher bank were all found to be advantageous in making a more efficient final recovery.

The most marked improvement in handling the diluted pulp from the rougher cells is shown in circuit *E*. The dilute pulp from the primary cells was thickened and thus reduced in volume before it entered the scavenger circuit. By reducing the volume of pulp, less cell capacity and smaller amounts of reagents were required to make the same recovery. Thickening the pulp also improved the froth production as part of the slimes in the thickener was removed in the overflow. More important than these advantages, however, the thickening step effected the development of a supporting medium of higher specific gravity, which was instrumental in improved flotation of the coarser particles. Results of this circuit are given in Table 6. The capacity of the cells was highest with this circuit and reagent and power costs were low.

TABLE 6.—*Operating Data for Thickening Circuit, Circuit E*

Product	Slurry, Tons per Hr.	Solids in Slurry, Per Cent	Solids, Tons per Hr.	Ash, Per Cent Wt.	Distribution Solids, Per Cent Wt.
Hydroseparator feed	165	5.03	8.30	29.89	100
Hydroseparator overflow	153	2.02	3.10	43.09	37.4
Cell Feed, hydro underflow[a]	26.7	19.5	5.20	21.75	62.8
Rougher-cell coal	10.4	38.3	3.98	11.28	47.9
Thickener overflow	7.2	1.12	0.08	57.87	1.0
Scavenger-cell feed	9.1	13.5	1.22	44.03	14.7
Scavenger-cell coal	1.5	29.4	0.43	22.5	5.2
Cell refuse	7.6	9.3	0.71	72.39	8.6
Coal output, rougher and scavenger	11.9	37.0	4.41	12.78	53.1

Consumption of Reagent

Oil	Lb. per Hr.	Lb. per Ton Coal	Cost, Cents per Ton Coal
Pine	1.97	0.45	3.71
Fuel	7.45	1.69	1.97

General Operating Data

Operation, 7 hr.
Number cells operated, 7; flotation time, 12.9[c]
Power,[b] kw-hr., 235; kw. per ton, 7.61.
Coal recovery, per cent feed, 85.0 per cent
Recovery of combustible, 94.7 per cent
Efficiency (sink and float), 96.6 per cent

[a] Hydro underflow with dilution water.
[b] Includes centrifugal filter.
[c] Based on initial volume of pulp and total cell volume.

The pulp density or ratio of solids to water in the feed to the flotation cells was found to be an important factor. Laboratory tests, substantiated by pilot-plant tests, showed that recovery of coal as well as the quality of the product dropped as the percentage of solids in the feed pulp to the primary cells increased. Furthermore, the high recoveries of coal in the first few minutes of contact time dropped off at the higher pulp densities. Densities of 17 to 20 per cent solids in the pulp were found to give good results in practice. Apparently crowding of the particles in the cells takes place at higher pulp densities, as explained by Crawford* resulting in less efficient froth production and at the same time mechanical entrapment of some of the refuse with the froth.

Since the density of the feed pulp to the primary cells should be below 20 per cent solids in order to get the most complete recoveries, the density of the tailings from the cells drops to an excessively low value after the first few minutes of contact time, owing to the removal of the large quantity of coal. Recovery of the remaining coal in the diluted pulp was found to be inefficient, since considerably more reagent and contact time was required to make a final cleanup. Since the finer-sized coal floats during the initial flotation periods, the larger particles are somewhat concentrated in the dilute tailings from the first few cells. Thickening of the pulp to approximately the original density before final flotation assisted the final recovery by producing a denser separating medium. Furthermore, the advantage of reducing the volume of the pulp is obtained so that less cell capacity and reagents are required.

DEWATERING EQUIPMENT

An 18 by 28-in. centrifugal filter of the solid bowl type was first tested to dewater the coal froth. Operating data in Table 7 show that the centrifuge dewatered approximately 3 tons of coal per hour from 60 per cent to 26 per cent moisture at a power consumption of 7.73 hp. A recovery of 98.5 per cent was made in the filter. As the solids lost in the effluent had an ash content of 45 per cent, the quality of the filtered coal product was improved. Difficulty with the operation of this filter was at first caused by slippage of the dewatered cake against the rotating bowl while it adhered to the spiral conveyor. This condition was overcome by roughing the bowl surface.

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* J. T. Crawford: Importance of Pulp Density, Particle Size and Feed Regulation in Flotation of Coal. *Trans.* A.I.M.E. (1936) **119**, 150.

A 1½ by 3-ft. vibrating screen was tested to determine whether part of the dewatering job could be done on this type of equipment. Data in Table 7 show that approximately 53 per cent of the feed tonnage was dewatered to 33.6 per cent moisture over a 35-mesh Ton-Cap screen cloth.

TABLE 7.—*Operating Data on Dewatering Equipment*

Data	18 by 28-inch Bowl Centrifugal Filter	1½ by 3-foot Vibrating Screen
Feed, tons per hr.	3.00	3.02
Moisture, per cent	61.9	55.3
Dry cake, tons per hr.	2.95	1.60
Moisture, per cent	26.0	33.6
Recovery solids, per cent	98.4	53.0
Effluent loss, tons per hr.	0.05	1.42
Moisture, per cent	98.72	67.3
Loss solids, effluent, per cent	1.6	47.0
Consumption of horsepower	10	0.5 (motor rating)
Screen mesh		35
Speed, r.p.m.	1,300	1,420

In order to dewater this fine material over a screen, a continuous band or cake approximately 1½ in. thick was formed in the pool at the feed end of the screen deck. The cake was then run up a pitch of 1 to 1½ in. per foot. Rapid formation of the cake in the pool and dewatering through the cake was assisted to some extent by strong water sprays on the screen deck. Increasing the frequency of vibration and the positive pitch on the deck helped to produce a drier cake. The dewatering performance on a screen could be considerably improved by thickening the coal froth ahead of the screen. A higher recovery over the screen would then be realized as well as a drier cake.

A dewatering circuit was designed with the vibrating screens in series with a centrifugal filter. All the coal froth was fed directly to the screens, which dewatered 50 to 60 per cent of the coal. The underflow from the screens was then sent to the centrifugal filter, where final recovery and dewatering of the coal was accomplished. Considerably less power and investment cost were required than if all the froth was dewatered by centrifugal filters.

SUMMARY

The flotation tests using the various circuits with a proper classified feed showed that a high recovery of coal took place in the first few minutes of contact time, followed by a more inefficient treatment of a dilute pulp throughout the remaining flotation circuit. Recleaning the froth from the last few cells by returning the froth to the head of the bank of cells was found advantageous. The maximum capacity of the flotation cells with a minimum of power and reagents was obtained by circuit *E*, in which two banks of rougher cells are used to utilize the high recovery possible in the first few minutes of contact time. The diluted pulp from the primary cells in this circuit was then thickened, with consequent reduction in volume, before being sent to the scavenger cells.

Dewatering of the coal froth to 26 per cent moisture was accomplished in a centrifugal filter at a power consumption of 2.5 to 3.0 kw. per ton of coal. Dewatering the coal froth to 33 per cent moisture was performed on a vibrating screen with losses of 40 to 50 per cent of the feed through the screen. Since considerably less power is required for operating the screen, a circuit employing both the screen and centrifuge was planned whereby the coal lost through the screen deck would be recovered and dewatered by the centrifuge.

All the economic factors in the recovery of fine anthracite from wash water by froth floation could not be definitely determined by the pilot plant. Operating costs for power and reagents were found to be 13 to 15¢ per ton of clean coal. Labor, maintenance, and amortization charges appeared to be sufficiently low to make the project appear economically feasible. One important problem, still being considered, is the complete drying of the product before shipment to facilitate unloading and handling by the consumer. If this step is found necessary, the cost will be considerably increased.

ACKNOWLEDGMENTS

The writers take this opportunity to thank Mr. R. E. Hobart and Mr. A. G. Frank, of the Lehigh Navigation Coal Company Inc., for their assistance during the construction and operation of the test plant. Grateful acknowledgment is given Mr. H. J. Daniels, formerly with the Denver Equipment Co., for his valuable suggestions and help during the experimental work. Thanks are also due the Denver Equipment Co., the Dorr Company, Allis Chalmers Mfg. Co., and the Bird Machine Co. for their cooperation.

DISCUSSION

(H. F. Hebley presiding)

J. W. WOOMER,* Wheeling, W. Va.—We are interested in the paper from the stream pollution angle. I wonder if the author would care to give us some thoughts on that aspect of the matter.

W. J. PARTON (author's reply).—One of the main reasons for our experimental work was to devise means to minimize stream pollution. Increased recovery of fine coal was, of course, another objective. The commercial plant we are now building will completely solve the stream-pollution problem from that colliery. Of course, we have to use a thickener in conjunction with the flotation plant, to remove the solids from the hydroseparator overflow and the cell tailings.

J. W. WOOMER.—Are you recirculating the water in the plant?

W. J. PARTON.—Yes. We contemplate taking the Bird centrifuge effluent and using it as dilution water in the conditioning tank. By recirculating this water we hope to economize on some of the reagents.

B. M. BIRD,* Columbus Ohio.—One use for froth flotation is to reduce sulphur in Pittsburgh coal. Everyone is making a fair percentage of extremely fine sizes. That the pyritic sulphur can be markedly reduced in those sizes by froth flotation has already been demonstrated. This idea can, of course, be extended to all sizes by deliberate grinding and floating of all the coal. I do not think we are ready for that step, but it is a definite out for reducing the pyritic sulphur in the Pittsburgh area. The idea strikes me as rather fantastic to grind coal down through perhaps 65 mesh and then to float it, but we may be forced into doing it.

I think a more promising method is to attack the problem from the point of view of removing the sulphur from the pig iron or during the cokemaking process. Personally, I should like to see the work that Dr. Powell, Mr. Thompson, and others did a good many years ago revived and reviewed carefully. The economic pressure was not on them in those days, and perhaps now that we are faced with this serious problem of sulphur reduction something can be done along the lines of their work.

D. H. DAVIS,† Pittsburgh, Pa.—I might add a little on froth flotation. I note that at the plant referred to by Messrs. Rupert and Parton, they were cleaning approximately six tenths of a ton per hour per cell. This checks pretty well with our experience in froth-flotation thickener underflow from cleaning plants handling bituminous coal. It will be noted, and this will answer Mr. Woomer's question, I believe, that a large percentage of the minus 200-mesh is removed from the feed to the flotation units by the hydroseparator. In order to recover the minus 200-mesh it is necessary to dewater it in such a manner that the material can be wasted without contamination of streams. The usual practice is to pump it to a dam for impounding and let the water evaporate or filter through the dam. At the Pittsburgh Coal Co. we filter the minus 200-mesh and mix it with coarse refuse for disposal.

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* Consulting Mining Engineer.

* Battelle Memorial Institute.

† Pittsburgh Coal Company.

A question has been raised about cleaning of the minus 200-mesh by froth flotation. As this material is mostly clay and is in a very finely divided state, the capacity of each cell in cleaning this material is extremely low, running approximately ¼ ton per hour per cell. Therefore rather a large plant is required to clean any appreciable tonnage of minus 200-mesh material.

Referring to Mr. Bird's remarks, I would say that everything should be done within practical limits to prevent the production of too much minus 200-mesh material, for once a mixture of coal and slate is 200-mesh or finer in size, it is usually uneconomical to try to recover the clean coal.

C. E. Locke,* Cambridge, Mass.—In two places the authors have made statements that are not clear to me. On page 182, in the last sentence of the second column, they speak of a hydroseparator overflow rate of 12 ft. per hour, and similarly in the last line of Table 2, they give the classifying rate in feet per hour. I am unable to understand how they can apply any rate of linear feet per hour in separation in a hydroseparator, because the only velocity that one can measure in such a device is the surface velocity of flow from the center to the circumference. To my mind this will mean very little, and as a matter of fact it is not an average because such a flow would be rapid near the center and decrease to a minimum close to the peripheral overflow.

My second difficulty is my inability to visualize the dewatering screen they describe at the bottom of the first column on page 188. They speak of a pool at the feed end, which makes me feel that the screen is flat at that point and then changes to a slight slope upward of 1 or 1½ in. per foot. The application of the strong water jets as an aid to dewatering is not clear.

J. W. Parton.—Dr. Locke is correct in questioning the overflow rate in feet per hour, since this is a calculated figure and not an actual measurement. The overflow rate, as we called it, was calculated by dividing the volume of feed in cubic feet per hour by the surface area of the hydroseparator tank. This relationship has been found convenient and reliable for use in determining the size of hydroseparator required for a given feed when the classifying rate was determined by a small test hydroseparator, as in our work. Tube classification rates can also be projected quite accurately into practice by using a correction factor for turbulescence.

The deck of the dewatering screen is not flat at the feed end, as Dr. Locke suggested. The whole deck surface is carried at the same upward pitch of 1 to 1½ in. per foot. At the low end of the deck a pool forms; the back of the screen frame serving as the back of the dam. Drainage from this pool is quite slow if a compact bed of solids forms against the screen deck. Accordingly, we found that strong water jets applied to the shallow pool tended to keep the layer of coal in a loose condition and enabled more rapid drainage from the pool.

C. Evans, Jr.,* Scranton, Pa.—Have you decided about what you are going to attempt to do with the very fine waste? You have a considerable quantity to dispose of. I do not believe we in the anthracite region have gone quite as far as they have in the Pittsburgh district in contemplating drying and disposing of it.

W. J. Parton.—We will dispose of the fine waste solids by first concentrating these solids in a large thickener along with the refuse from the cells and then pumping the thickened sludge to impounding dams.

C. Evans, Jr.—Have you a surface free from stream flow on which you can put those impounding dams?

W. J. Parton.—We are planning impounding dams on old refuse banks and hope to get some filtration through the banks. But we are counting mainly on the concentration in the thickener and do not expect to pump a large quantity of water with the solids to the dam.

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* Professor of Mining and Ore Dressing, Massachusetts Institute of Technology.

* The Hudson Coal Company.

C. Evans, Jr.—Pioneering of that kind is a big problem.

W. J. Parton.—Particularly in our field.

Member.—Have you any figures on the complete drying of the product?

W. J. Parton.—We have studied the subject of heat drying of the product but we are not in a position to give any figures at this time.

Member.—On what specific gravity is efficiency of sink and float based in Table 3?

W. J. Parton.—I can't quite recall what that gravity was, since I picked the theoretical recovery figure off a curve. However, I think it corresponded to a 1.9 gravity.

J. Griffen,* Pittsburgh, Pa.—What is meant by efficiency by sink and float?

W. J. Parton.—The Bureau of Mines uses the recovery by sink-and-float data as the basis for comparing the actual recovery to determine operating cleaner efficiency.

J. Griffen.—Of the same ash content?

W. J. Parton.—Yes.

J. Griffen.—As 100 per cent?

W. J. Parton.—That is right. The actual recovery of coal divided by the theoretical recovery at same ash content from sink-and-float data times one hundred gives the cleaner efficiency.

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* McNally Pittsburg Manufacturing Co.

A New Criterion for the Clinkering Characteristics of Coal Ash

By Raymond S. Weimer,* Member A.I.M.E.

(St. Louis Meeting, September 1942)

Of the several characteristics of a coal that influence its suitability for use in a given piece of burning equipment, the clinkering and slagging characteristics of the ash of the coal are highly important. They determine the maximum rate at which the coal may be burned on a stoker or in pulverized form without difficulty from damage to metal parts, from erosion of refractory walls, or from stoppage of gas passages by accumulation on the tubes. They also determine the minimum rate at which a slag-tap furnace may be operated without difficulty in removal of the ash in fluid form.

Need for a Better Criterion

In common with other producers and users of coal, the Northern Illinois Coal Corporation has long used the available methods for the measurement of the clinkering characteristics of coal ash, chemical composition and fusion temperature of the ash, as criteria for the selection of coal that would suit the particular requirements of its customers. These two available criteria have greatly helped in the problem of selection but it is unnecessary to dwell upon the fact that they are not completely adequate for the purpose.

The composition of the ash and its clinkering tendency vary widely over the company's mining area. In a certain period, some years ago, an unusual amount of severe clinkering of the ash of coal shipped made extremely acute the need for a better criterion of the clinkering that might be expected. It was also desirable that the criterion result from a reasonably rapid method, so that the clinkering characteristics of the coal would be known before shipment.

Complaints on the severe clinkering of the ash came from only a fraction of the output of the mine but the occurrence of the clinkering seemed quite unpredictable. Attempts to relate the chemical composition of the ash and the fusion temperatures to the clinkering were unsuccessful. Coal from the area of the mine in which the fusion temperature of the ash was the lowest gave the least trouble from clinkering. Chemical analysis of the ash from that section giving the most trouble showed high contents of iron and low contents of silica, but frequently these were not greatly different from those of coals that gave little trouble from clinker.

Aside from the clinkering difficulty, the coal was quite satisfactory. The ash content was normal and high rates of burning could be obtained, but again and again such severe clinkering would occur that the furnace would be shut down because of the accumulation of clinker. Examination of the clinker from the customers' plants showed that the clinkers that adhered to the grates and fouled the fuel beds were hard, flat, dense, and usually dark in color. Coal that was satisfactory gave clinkers that were softer, bulky rather than flat, of low density, and usually lighter in color.

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* General Superintendent, Wilmington Mine, Northern Illinois Coal Corporation, Wilmington, Ill. Died April 17, 1943.

DENSITY SUGGESTED AS CRITERION

Two measures of the properties of the clinker were thus suggested, color and density. Although the color did seem to bear a relation to the degree of difficulty encountered, it was not wholly reliable and could not conveniently be expressed in a numerical scale. Density, on the other hand, could be measured accurately and expressed in numerical values representing the true or apparent specific gravity. An opportunity was offered also for comparison of the density of clinkers formed under standard conditions in a laboratory test furnace with those formed in the furnaces in customers' plants. It was conceivable that desirable limits of the density could be set up for different customers and, by knowledge of the density of the clinkers formed from coal in different parts of the mine, shipments could be so controlled that each customer would receive coal that would meet the requirements of his particular conditions.

Extensive experimental investigation, in which the true and apparent specific gravity of the clinkers of coals burned in several types of test furnaces were measured, revealed that the specific gravities were reproducible in repeat tests and that they were related to the degree of difficulty experienced by the customers in their plants. It was found also that certain materials could be added to the coal to modify the clinkering characteristics of the ash and that the change in the specific gravity resulting from the additions offered a means for control of the amount of material required to effect the desired result.

After considerable experience with the method, a program of research was sponsored by the Northern Illinois Coal Corporation at Battelle Memorial Institute, in which the method was checked under more carefully controlled and measured conditions than had been used at the mine laboratory. Many of the data included in this paper are the result of that program.

This paper describes the method of formation of clinkers in the laboratory, the methods for measurement of the specific gravity of the clinkers, the relation of the specific gravity to the rate of burning and fuel-bed temperature, to results obtained on industrial stokers, and to other measurable characteristics of the ash, the effect of blending of coals on the specific gravity, the effect of addition of materials to modify the specific gravity, and the method used at the mine in normal operation to control the clinkering characteristics of the coal shipped.

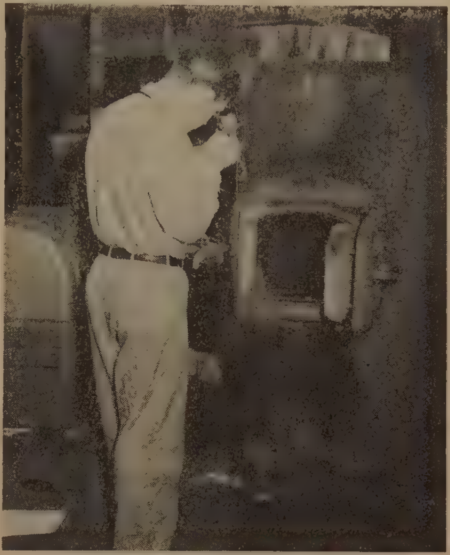

FIG. 1.—TEST FURNACE USED AT MINE LABORATORY.

EQUIPMENT AND METHOD OF TESTING

First attempts to produce clinkers whose specific gravity could be measured were made on a small laboratory scale, using a few grams or pounds of coal. Although this appeared desirable, it is obvious that the work required to obtain a small sample that is representative of a large lot of coal may be greater than that to burn directly a larger amount of coal. Furthermore, a simulative test on a larger scale requires less extrapolation to industrial furnaces than does the extremely small-scale test. Several furnaces have been used; they have varied from a small heating stove, fired by hand but with a forced-draft blower, to an underfeed stoker burning 150 lb. of coal per hour.

Fig. 1 shows the test furnace that is now in regular use at the laboratory of the mine. A similar furnace was used in the investigation at Battelle. A steel shell is lined with insulating refractory brick and firebrick, to give a combustion chamber 33 in. high and 27 in. in diameter. It is fired with a residential-type screw underfeed stoker with an 11-in. diameter retort; it has a feed rate of 30 to 60 lb. of coal per hour. The dampers for control of the air supplied by the stoker fan and for control of the stack draft are set manually.

The standard procedure in each burning test is as follows: The hopper, feed screw, retort, and hearth are cleaned of any coal, ash, or clinker from previous tests. One hundred pounds of 0 to 1-in. coal are placed in the hopper and coal is fed to the level of the top of the retort. The coal is kindled and the stoker is operated continuously at the feed rate of 33 lb. per hour. In the tests conducted at Battelle, the air was supplied by a separate fan through a duct in which the rate of air supply was measured by means of the pressure drop across a thin-plate orifice; for standard tests, the rate of air supply was adjusted to give approximately 20 per cent excess air. In the routine tests at the mine, the air is not measured but is controlled by adjustment of the damper on the stoker to give a normal type of fuel bed.

The operation of the stoker is continued until the last of the coal is fed into the retort and the fuel bed burned down to the level of the top of the retort. The stoker is then shut off, the bed allowed to burn out and to cool. The burning test thus requires about three hours.

The residue is removed from the furnace and separated into unburned coke, fine ash, and clinker larger than ¼ in., and the weight of each is recorded. No material that is not fused is considered as clinker. If the amount of such material is large, the test is not considered reliable.

The temperature in various parts of the fuel bed is read with an optical pyrometer at 15-min. intervals throughout the test, recorded, and averaged. As discussed later, such average temperatures have been found to bear a significant relation to the rate of burning and to the specific gravity of the clinkers, although it is recognized that a wide range of temperatures can be observed in the fuel bed of an underfeed stoker, dependent on the point in the bed on which the optical pyrometer is sighted.

MEASUREMENT OF SPECIFIC GRAVITIES

The apparent specific gravity of the clinkers is determined separately on each of the two or three parts into which the lot of clinker is divided. The determination is made in the usual way, by measurement of the weight of water displaced by the given weight of clinker. This is conveniently done in a 2-liter graduated cylinder.

The true specific gravity is determined by a modification of the A.S.T.M. Standard Method for Coke, D-167-24, in which a stoppered glass vial is used instead of the Hogarth specific-gravity bottle. A 3-gram sample of clinker ground to pass the 200-mesh sieve is covered with water in a 12-ml. glass vial. The air is removed by boiling for one hour, the bottle is cooled and filled with distilled water, the stopper is put in place and the vial and contents are weighed.

The true specific gravity is calculated as follows:

$$\text{True sp. gr.} = \frac{W}{W - (W' - P)}$$

where

W = weight of dry sample, grams,
W' = weight of vial + stopper + dry sample + water to fill vial, grams,
P = weight of vial + stopper + water to fill vial, grams.

REPRODUCIBILITY OF RESULTS

Table 1 presents the data from four burning tests on one coal as made during the investigation at Battelle. Tests 1 and 2 were made at the same rate of burning but with different rates of supply of air, corresponding to 18 and 59 per cent excess. The mean fuel-bed temperature should have been lower with the higher excess of air and apparently this is shown by the somewhat smaller percentage of clinker removed. The apparent specific gravities of the clinker were, however, practically identical, which indicates that variations of the rate of air supply, within the range shown, made no difference in the results.

TABLE 1.—*Data from Several Stoker-furnace Tests, Showing Reproducibility of Apparent Specific Gravity*

Test No.	1	2	3	4
Average burning rate, lb. per hr.	34	32	46	46
Weight loss, lb.	100	100	150	150
Burning time, hr.	2.95	3.12	3.25	3.25
Air rate, lb. per hr.	342	432	470	463
Air per lb. weight loss, lb.	10.1	13.5	10.2	10.0
Excess air, per cent[a]	18	59	23	26
Mean stack temperature, deg. F.	1380	1380	1540	1420
Maximum stack temperature, deg. F.	1500	1585	1720	1670
Weight of +¼-in. clinker, oz.	20	16	28	29
Clinker, per cent of ash released	33	26	31	32
Apparent specific gravity of clinker	2.53	2.52	2.63	2.61

[a] Calculated from flue-gas analysis, disregarding initial 15-min. interval of test period.

Tests 3 and 4 were made at a higher burning rate but with approximately 20 per cent excess of air. The percentage of ash fused to clinker was not greater than in test 1 but the apparent specific gravities were significantly higher than those of tests 1 and 2.

Table 2 presents the results of burning tests conducted at the mine laboratory on two coals. In addition to the apparent specific gravity of the clinker, the true specific gravity, the fuel-bed temperatures, and the chemical composition of the ash of the coal and of the clinkers are given. Significant facts from the tabulation are that the chemical composition of the clinkers did not vary appreciably from that of the ash of the coal, and that the true specific gravity of the clinkers did not vary appreciably, although the apparent specific gravity increased as the fuel-bed temperature increased with increase of burning rate. A difference in the chemical

TABLE 2.—*Data from Stoker-furnace Tests of Two Coals at Different Feed Rates*
SHOWING CONSTANCY OF TRUE SPECIFIC GRAVITY AND CHEMICAL COMPOSITION BETWEEN COAL-ASH ANALYSIS AND CLINKERS

	Original Coal	Coal 173		
Lb. per hr.		30	53	68
Ash, dry basis, per cent	8.25			
Heating value, dry basis, B.t.u. per lb.	13,091			
True sp. gr.		2.96	2.93	2.90
Apparent sp. gr.		1.62	1.97	2.01
Average fuel-bed temp., deg. F.		2519	2669	2708
Composition, per cent:	ASH	CLINKERS		
SiO_2	39.05	40.25	41.01	39.56
Al_2O_3	22.65	24.16	23.78	23.87
Fe_2O_3	32.93	31.05	30.38	31.35
CaO	1.09	1.31	1.14	1.29
MgO	1.03	1.12	1.07	1.16

	Original Coal	Coal 1029			
Lb. per hr.		33	34	48	68
Ash, dry basis, per cent	5.62				
Heating value, dry basis, B.t.u. per lb.	13,579				
True sp. gr.		3.18	3.19	3.19	3.15
Apparent sp. gr.		2.12	2.04	2.28	2.42
Average fuel-bed temp., deg. F.		2433	2419	2653	2815
Composition, per cent:	ASH	CLINKERS			
SiO_2	31.05	31.92	31.95	32.35	31.80
Al_2O_3	21.09	20.50	20.56	20.68	19.91
Fe_2O_3	36.20	37.83	37.00	37.68	37.80
CaO	5.24	5.03	5.09	5.43	5.35
MgO	0.77	0.49	0.57	0.53	0.61

composition of the clinkers and of the ash would not be entirely unexpected, as a stratification and separation of constituents of widely varying composition and fusibility frequently occurs in fuel beds. The constancy of composition for clinkers and coal ash for these coals indicates the absence of stratification and a general homogeneity of the original ash of the coal.

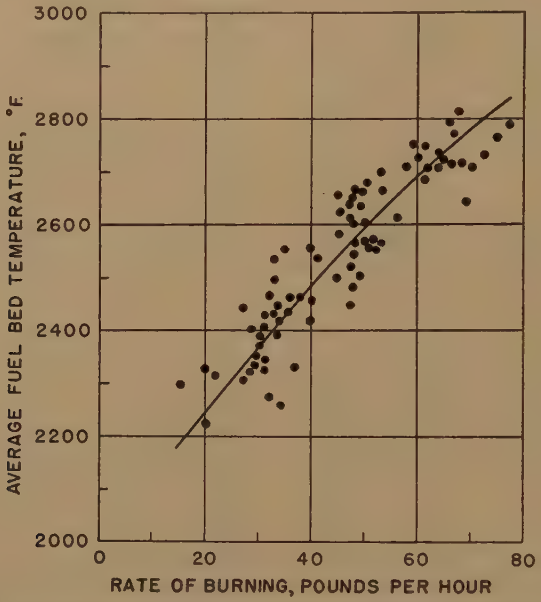

Fig. 2.—Relation of rate of burning in stoker to average temperature of fuel bed.

Relation of Fuel-bed Temperature to Apparent Specific Gravity of Clinkers

The temperature in the fuel bed of an underfeed stoker, at the same rate of feeding coal, will be expected to vary, even with the same excess of air, because of variation in the calorific value of the coal. Also, it is impossible to maintain a constant rate of feed for different coals because the stoker screws feed by volume rather than by weight, and the bulk densities of coals vary because of differences in their size distribution. For comparison of results on different coals, therefore, it is desirable to be able to correct to a standard reference temperature the value of apparent specific gravity obtained on a test.

Furthermore, temperatures in the fuel beds of industrial stokers vary from those in the laboratory test furnace, because of differences in the burning rates, in the size of the furnace, and in the temperature of the walls of the furnace. For this reason, also, it is desirable to know the relation between the apparent specific gravity and the temperature so that the laboratory results may be extrapolated properly to industrial furnace practice.

Fig. 2 shows the relation of the average fuel-bed temperature in the test furnaces to the rate of burning. As previously noted, the average fuel-bed temperature cannot be measured with accuracy because of the large variations in temperature in various parts of the bed and because of the rapidly changing conditions in the bed. For these reasons, temperatures for a given rate of burning vary over a rather wide range but there is a significant relation between the temperature and the rate.

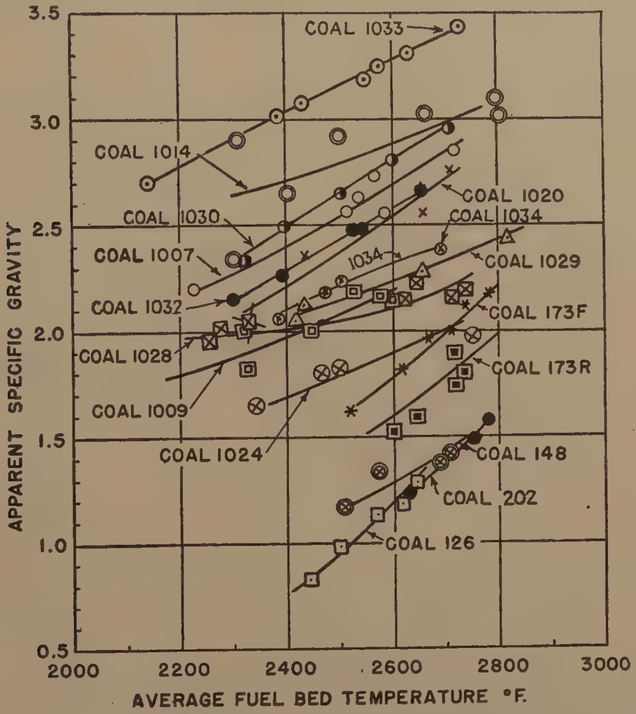

Fig. 3.—Relation of average temperature of fuel bed to apparent specific gravity of clinkers for several typical coals.

Fig. 3 shows the relation between the apparent specific gravities of clinkers formed in several test furnaces and the average fuel-bed temperature for 12 different coals. Four to eight burning tests were made on each coal. Despite the wide range of apparent specific gravities of the clinkers formed by the different coals, there is a marked similarity in the slopes of the curves. A summation of the results of many tests has shown that the rate of change of the apparent specific gravity with temperature is 0.0016 per degree. For the clinkers of higher apparent specific gravity, the rate of change is 0.0014 per degree; for those of lower gravity, the rate is 0.0018 per degree.

Using these values, the apparent specific gravity at any temperature may be calculated from the values for gravity and fuel-bed temperature found in a given test. A temperature of 2600°F. has been chosen as a reference point and observed values have been calculated to this point for most of the data presented in this paper.

Relation of True and Apparent Specific Gravity

The true specific gravity of a given coal ash or clinker should have a unique value that is independent of the temperature at which the ash or clinker is heated, unless the mineralogical composition of the material is changed by the heating. The data on two coals given in Table 2 showed that the changes in true specific gravity with rate of burning and fuel-bed temperature were less than those in the apparent specific gravity.

The apparent specific gravity is less than the true specific gravity because of the inclusion within the fused mass of closed cells that are not filled with water when the clinker is immersed in water. As the clinker is subjected to higher temperatures, its viscosity decreases, and the closed cells decrease in size as the enclosed gas escapes or as the surface tension can draw the mass together. Thus, the apparent specific gravity increases with increase of temperature, as has been shown.

Fig. 4 shows the relation between the apparent specific gravity, corrected to 2600°F. by the method previously explained, and the true specific gravity. A large number of determinations on clinkers from many coals at different temperatures are included in the figure. As would be expected, the curve approaches the limit of equality of true and apparent specific gravity. A surprisingly close relation between the apparent and true specific gravities is indicated.

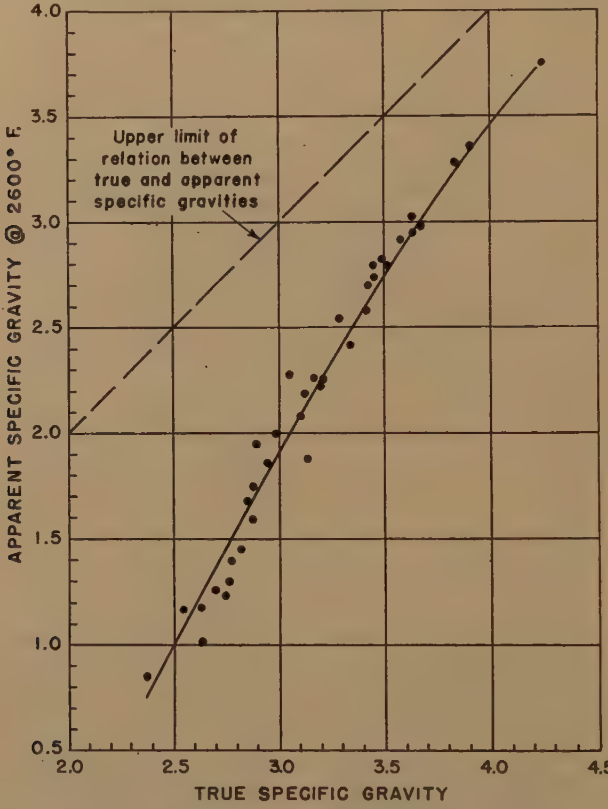

FIG. 4.—RELATION OF TRUE SPECIFIC GRAVITY TO APPARENT SPECIFIC GRAVITY AT 2600°F.

CORRELATION OF TEST DATA WITH RESULTS ON INDUSTRIAL FURNACES

For this new criterion of the clinkering characteristics of coal ash to be proved applicable to the selection of coal for industrial furnaces, it is necessary to establish a relation between the laboratory data and those obtained in operating furnaces. It is also necessary to establish the range of apparent specific gravities within which satisfactory operation occurs.

"Satisfactory" and "unsatisfactory" are rather loose terms as applied to the performance of a coal or the clinkering characteristics of an ash, but often no more definite explanation is obtained by the producer from the customer. When expressed in more definite terms, such phrases as "sticking clinker," "fouled fuel bed," and "slow heat release" often are used. A sticking clinker is one that becomes very fluid, runs to the grate to shut off the air, and sticks to the grates and walls. It will limit the upper rate at which the fuel can be burned and thus limit the output of the boiler. A fouled fuel bed is one in which the ash accumulates in the bed, increasing the resistance to the flow of air. The result usually is the same as expressed by the phrase "slow heat release" and the characteristics of the coal may also be described as "too much ash." By increasing the resistance of the bed to the flow of the air, or by shielding the fuel from contact with the air, the desired rate of heat release is not obtained. In such fuel beds, the ash remains voluminous and often fluffy. It occupies much volume. A coal with the same or even higher content of ash by weight would be satisfactory if it made a denser clinker that did not shut off the flow of air or shield the fuel from the air.

Table 3 presents a comparison of the specific gravities of clinkers produced in the standard laboratory control tests on coal sampled at the mine, in the same type of tests on coal sampled at the user's plant, and on clinkers taken from the plant furnace, together with notes on the results that the user obtained with the coal. The tabulation includes coals whose ash contents were 4.7 to 11.4 per cent.

Comparison of the true specific gravities of the clinkers formed in the test and in the operating furnaces shows that there was good agreement between the values despite the probable differences in the temperatures in the two furnaces. This is in accord with the reasoning that the true specific

TABLE 3.—*Comparison of Specific Gravities of Clinkers from Tests of Samples at the Mine and at the Users' Plant, and of Clinkers from Users' Furnace*

At Mine		From User's Plant					Type of Equipment and Service	Results
Standard Control Test					Furnace			
Coal	Clinker	Coal	Clinker, Specific Gravity					
Ash, Dry Basis, Per Cent	Apparent Sp. Gr.[a]	Ash, Dry Basis, Per Cent	True	Apparent[a]	Apparent	True		
10.9	1.7	11.2	2.9	1.7	2.3	2.7	Steam underfeed heating	Satisfactory
12.3	1.5	10.3	2.8	1.6	1.2	2.9	Underfeed steam	Slow heat release
6.1	3.1				2.9		Underfeed power	Clinker sticking
6.6	3.1				3.3	3.9	Incline grate heating	Clinker sticking
		7.6	3.7	2.9	3.3	4.1	Underfeed heating	Clinker sticking
					2.3	3.5	Hand fire heating	Satisfactory
		4.7	3.5	2.3[b]	2.7	3.5	Underfeed heating	Clinker sticking
					1.7		Hand fire power	Satisfactory
		11.4	3.1	2.0	1.8	3.1	Chain-grate heating	Satisfactory
		10.6	3.0	2.2	2.6	2.9	Incline grate	Satisfactory
		9.1	2.7	1.4	1.3		Locomotive	Slow heat release Fouled fuel bed
		7.0	3.0	1.9	1.7		Locomotive power	Satisfactory
		10.1	2.8	1.5	1.8		Underfeed steam	Slow-heat release
					2.2		Underfeed	Satisfactory

[a] Adjusted to 2600°F.
[b] Temperature not observed.

gravity is a unique characteristic of a coal ash of given chemical and mineralogical composition.

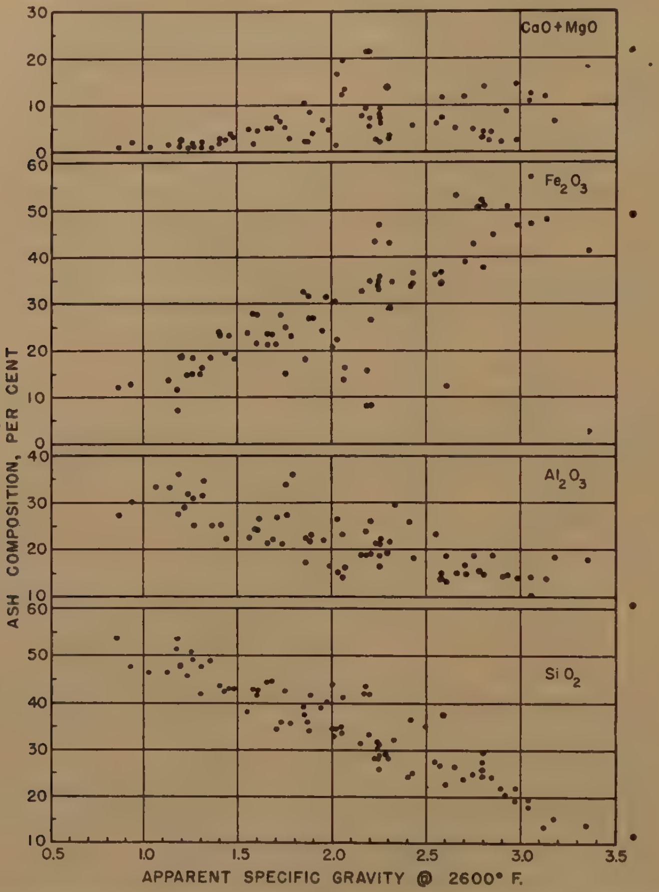

FIG. 5.—RELATION OF APPARENT SPECIFIC GRAVITY OF CLINKER TO CHEMICAL COMPOSITION OF ASH

The agreement of the apparent specific gravities obtained in the test furnace with those in the operating furnace is not as close as for the true specific gravity. This is expected because of the difference in temperature conditions. Frequently, however, the agreement is close and they are always of similar order.

Coals that are classified as giving a sticky clinker gave clinkers of relatively high apparent specific gravity—2.7 to 3.3. Those that were rated as satisfactory gave clinkers of medium apparent specific gravity, 1.7 to 2.6. One that was rated as giving a slow heat release had a moderately high specific gravity, 1.8, but the other two, which were unsatisfactory, for a similar reason had relatively low specific gravities, 1.2 to 1.3.

The indications from these data, which are supported by many other data obtained in the years of experience with this method, are that for most satisfactory performance the clinker obtained in the standard test furnace should have an apparent specific gravity of about 2.0 to 2.5. On coals whose clinker has higher apparent specific gravities, complaints will be received on excessive clinkering; on coals whose clinker has lower gravities, complaints will be received on slow heat release and excessive amounts of ash.

Relation of Apparent Specific Gravity of Clinker to Chemical Composition and Ash-fusion Temperatures

During the course of the development of this new criterion for the clinkering characteristics of ash, data have been obtained, for most of the coals tested, on the chemical composition and the fusion temperatures of the ash.

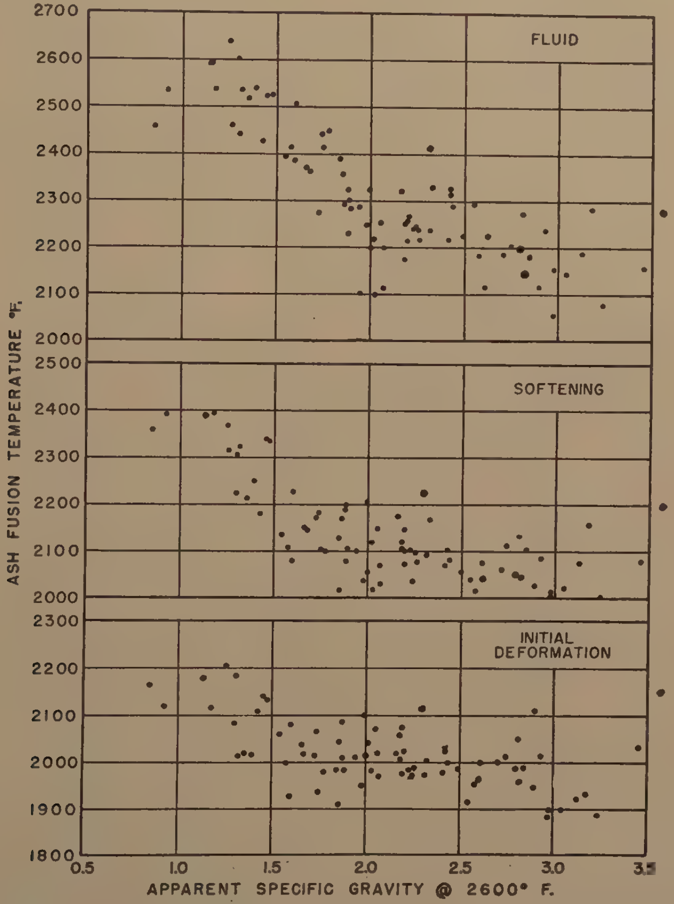

Fig. 6.—Relation of apparent specific gravity of clinker to fusion temperature of ash.

Fig. 5 shows the relation between the contents of silica, alumina, ferric oxide, and calcium plus magnesium oxides and the apparent specific gravities of the clinkers, corrected to 2600°F. About one-half of the coals included in this plot are from the Illinois No. 2 seam, in which the Wilmington mine of the Northern Illinois Coal Corporation is working. Others are from the Illinois Nos. 5 and 6 seams, and from seams in Indiana, West Virginia, and Kentucky. Definite trends are shown.

The apparent specific gravity increases as the silica and alumina contents decrease, and it increases as the ferric oxide and lime and magnesium oxides increase. The dispersion of values for any one gravity is high and it is apparent that the apparent specific gravity of the clinker cannot be closely predicted from the chemical composition.

TABLE 4.—*Composition and Fusion Temperatures of Ash and Apparent Specific Gravity of Clinker*

Sample No.	Coal Seam	Ash, Dry Basis, Per Cent	Fusion Temperatures, Deg. F.			Chemical Composition of Ash, Per Cent				Apparent Sp. Gr. at 2600°F.
			Initial Deformation	Softening	Fluid	SiO_2	Al_2O_3	Fe_2O_3	MgO + CaO	
126	Ind. 5	10.3	2120	2390	2540	53.8	27.4	11.9	1.4	1.17
148	Ind. 5	8.7	1850	2320	2460	49.2	25.5	18.6	1.3	1.26
121	Ind. 5	9.9	2020	2250	2540	43.8	25.3	24.1	3.0	1.39
893	Ill. 2	11.5	2140	2340	2530	43.0	29.9	18.3	4.4	1.45
173	Ind. 6	14.7	1910	2020	2360	42.8	24.5	28.0	2.0	1.59
1023	Ill. 6	10.5	2020	2150	2370	44.5	22.2	23.7	5.0	1.67
1025	Ill. 2	13.8	2440	2190	2070	39.4	27.6	25.1	5.3	1.74
173	Ind. 6	8.3	1980	2120	2390	39.1	22.7	32.9	2.1	1.85
1015	Ind. 5	10.6	2070	2190	2300	34.4	30.0	31.9	2.2	1.88
1024	Ill. 2	8.9	2300	2100	2010	39.0	22.1	24.4	6.7	1.94
767	Ill. 6	7.7	2100	2210	2330	34.8	23.1	19.9	10.2	1.99
1009	Ill. 2	7.5	1980	2100	2240	28.1	21.7	43.5	2.5	2.23
1020	Ill. 2	4.6	1920	2040	2290	27.3	23.7	36.7	5.7	2.54
935	Ill. 2	6.4	1950	2020	2180	26.9	15.2	37.1	11.1	2.58
1007	Ill. 2	7.8	2000	2070	2180	23.5	15.2	39.2	11.6	2.70
1008	Cherokee, Kan.	7.5	2010	2110	2210	24.8	19.1	43.0	4.3	2.74
1014	Ill. 2	6.1	1900	2020	2150	19.4	14.2	47.1	10.4	3.04
44	Ill. 2	7.7	2090	2210	2280	13.3	15.5	59.0	6.6	3.75
12/13	Ind. 5	10.0				32.2	23.3	30.9	5.4	1.94
119-a	Ind. 5	10.2	2170	2360	2460	53.8	27.4	11.9	1.4	0.85
119-b	Ind. 5	16.8	2580	2680	2810	51.2	36.2	7.4	1.5	1.17
119-c	Ind. 5	8.9	2190	2310	2440	42.1	34.9	16.5	1.3	1.30
119-d	Ind. 5	10.0				46.4	33.5	13.9	1.3	1.02
119-e	Ind. 5	10.1				37.7	24.3	26.1	4.4	1.64
119-f	Ind. 5	11.2				43.5	27.7	19.3	3.2	1.34
119-g	Ind. 5	9.5	2020	2080	2320	36.9	28.7	24.1	3.5	1.75
119-h	Ind. 5	10.0				35.8	25.8	26.6	4.4	1.75
119-i	Ind. 5	10.0				39.3	28.4	22.3	3.4	1.53
119-j	Ind. 5	10.0				42.8	30.9	18.1	2.4	1.37

Fig. 6 shows the relation of the apparent specific gravity to the initial deformation, softening, and fluid temperatures of the clinkers as determined by the A.S.T.M. Standard cone test. Again, there is a general trend toward decrease of the fusion temperatures as the apparent specific gravity increases but the spread is large at any one gravity. A wide range of gravities is obtained for clinkers whose softening temperatures fall between 2000° and 2100°F.

Table 4 presents the fusion temperatures, chemical composition, and the apparent specific gravities of a part of the samples presented in Figs. 5 and 6.*

MODIFICATION OF APPARENT SPECIFIC GRAVITY AND CLINKERING CHARACTERISTICS

Although the determination of the apparent specific gravity of the clinker from a coal affords a means for the selection of coal suited to a particular plant, yet it was found that many parts of the mine yielded coal whose ash formed a clinker of such high apparent specific gravity that it was not suitable for use by many customers. Some means for control of the apparent specific gravity of the clinker, and thus of its suitability, were sought. Two methods, blending of two or more coals and the use of additional materials, have been tried and found satisfactory.

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* Editor's Note: An attempt was made to include a complete tabulation of the data of Figs. 5 and 6, for the benefit of those who might like to make a more detailed study of these values. Because of the untimely death of the author, it has been impossible to find the complete data used for the figures.

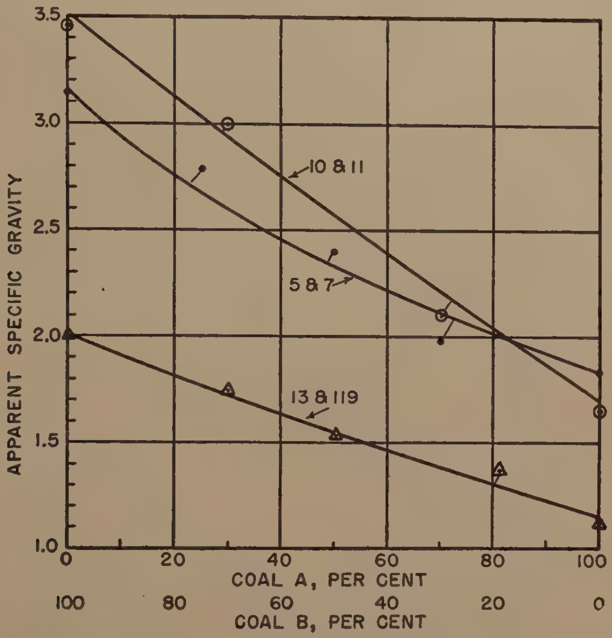


FIG. 7.—RELATION OF APPARENT SPECIFIC GRAVITY OF CLINKER TO RATIO OF BLENDS OF TWO COALS.

Modification by Blending

Fig. 7 shows three examples of the variation in the apparent specific gravity of the clinker as two coals are mixed in different proportions. The curves are slightly concave upward; that is, the addition of a coal whose ash forms a clinker of the lower specific gravity decreases the specific gravity of the higher slightly more than proportionately. Where two coals that form clinkers of widely different apparent specific gravity are passed through one preparation plant, it may often be possible so to blend the coals as to enable the successful use in a plant where the one could not otherwise have been used. The use of the laboratory testing furnace enables the selection of the proper ratio for blending.

Modification by Cleaning

That cleaning of the coal, or even screening, will frequently change the clinkering characteristics of the ash markedly is well known. Four examples of the effect of cleaning on the apparent specific gravity of the clinker from the standard test are given in Table 5.

TABLE 5.—*Effect of Cleaning on Apparent Specific Gravity of Clinker from Standard Test*

Coal No.	Condition	Ash Content, Per Cent	Apparent Sp. Gr. at 2600°F.
165-TR	As mined	10.9	1.35
165-TF	Cleaned	10.2	1.19
165-BR	As mined	11.0	2.37
165-BF	Cleaned	9.3	1.67
167-R	As mined	11.7	2.08
167-F	Cleaned	10.4	1.85
173-R	As mined	15.2	1.60
173-F	Cleaned	7.9	1.85

Cleaning of three of the four coals caused a moderate decrease in their ash content and a decrease in the apparent specific gravity of the clinker. The reduction in the ash content of the fourth coal, 173, was more marked and the specific gravity of the clinker was increased rather than decreased.

The volume of the ash may be increased although the actual weight is decreased.

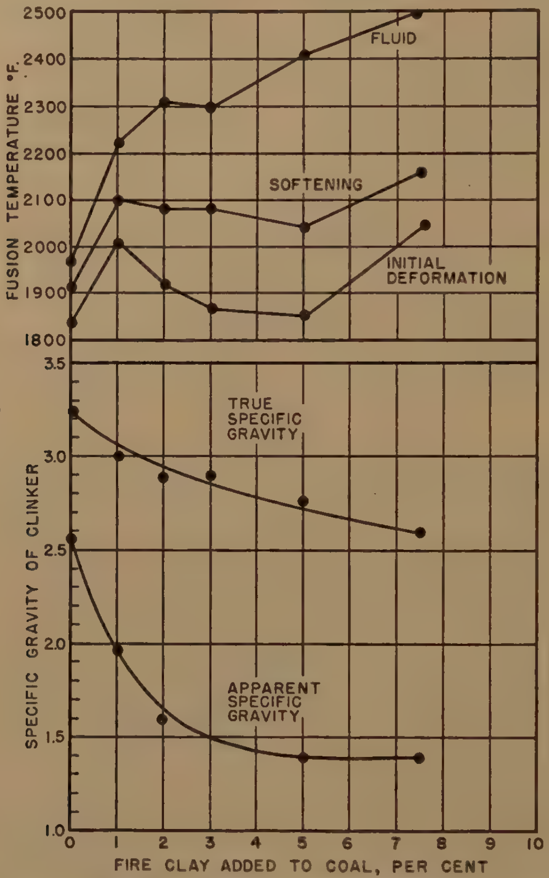


Fig. 8.—Relation of true and apparent specific gravities of clinkers and of fusion temperatures of ash to additions of fire clay to coal A.

The direction in the change of the specific gravity of the clinker undoubtedly depends on the nature of the ash-forming material that is removed from the coal. If the reduction is in the silica and alumina contents, the specific gravity will be increased; if pyrite or lime, or both, is removed, the specific gravity will undoubtedly be decreased.

If the specific gravity of the clinker is materially reduced by cleaning, complaints may be received on the amount of ash, whereas none were received before cleaning.

Modification by Additions to Coal

In the search for methods for the modification of the clinkering characteristics of the ash of the coal being produced at the Wilmington mine, trials were made of the addition of various materials to the coal. These materials included silica, bauxite, fire clay, and lime. Although it may appear incongruous to add ash-forming materials to coal after the removal of such materials by cleaning, it is justified if a useful result is obtained. Because the coal produced has an ash content of 4 to 7 per cent after cleaning, the addition of even 4 to 5 per cent of an ash-forming material would not increase the ash content above that of many coals sold on the same markets.

Fig. 8 shows the relation of the true and apparent specific gravities and the fusion temperatures of the ash of one coal from the Illinois No. 2 seam to the amount of fire clay added to the coal. The characteristics of the coal and of the ash were as follows:

PROXIMATE ANALYSIS, DRY BASIS, PER CENT	
Ash	4.5
Volatile matter	40.5
Fixed carbon	55.0
Total	100.0
Sulphur	1.3
Calorific value, B.t.u. per lb	13,610

ASH COMPOSITION, PER CENT	
Silica	27.1
Alumina	13.5
Ferric oxide	25.7
Calcium oxide	12.6
Magnesium oxide	2.2
Sulphur trioxide	10.7
Undetermined	8.2
Total	100.0

The burning tests were made at Battelle under carefully controlled conditions, at a rate of burning of 31 to 33 lb. of coal per hour. A total of 100 lb. of coal was used for each test. The determinations of the fusion temperatures were made on the clinker resulting from the burning tests.

Both the true and apparent specific gravities of the clinkers decreased with increasing amounts of fire-clay additions. The decrease in the apparent specific gravity was greater than that of the true specific gravity. The changes in the initial deformation and softening temperatures were not pronounced until additions of more than 5 per cent were made; then they increased. The fluid temperature of the clinker increased to a more marked degree than did the other fusion temperatures; this fact has been noted by other investigators. The data clearly demonstrate that the criterion of the apparent specific gravity is more reliable as a guide to the addition of the fire clay than are the fusion temperatures.

Fig. 9 shows similarly the relation of the specific gravities and fusion temperatures to the amounts of silica added to another coal from the Illinois No. 2 seam. The characteristics of the coal and its ash follow:

PROXIMATE ANALYSIS, DRY BASIS, PER CENT	
Ash	6.4
Volatile matter	45.3
Fixed carbon	48.3
Total	100.0
Sulphur	2.8
Calorific value, B.t.u. per lb	13,595

ASH COMPOSITION, PER CENT	
Silica	26.9
Alumina	15.2
Ferric oxide	37.1
Calcium oxide	10.2
Magnesium oxide	0.9
Alkalies	3.0
Sulphur trioxide	5.9
Undetermined	0.8
Total	100.0

These burning tests were made at the mine under less carefully controlled conditions than those made at Battelle. However, the fuel-bed temperatures were recorded and the apparent specific gravities are corrected to a temperature of 2600°F. The fuel-bed temperatures observed ranged from 2625° to 2515°.

The true and apparent specific gravities were decreased to a like extent with the additions of sand to this coal up to an addition of 4.2 per cent. A further addition to 5.6 per cent caused a slight increase in the gravities.

Fig. 10 shows similar data for the same coal but with the addition of limestone. The averages of the observed fuel-bed temperatures ranged from 2625° for the untreated coal to 2480° for the coal with the highest percentage addition of limestone. Up to about 3 per cent, the addition of limestone increased the apparent specific gravity of the clinker and it remained constant with increasing additions up to the maximum of 5.6 per cent used. Despite the more pronounced clinkering of the ash as shown by the increase in the specific gravity, the fusion temperatures did not decrease as might have been expected but, instead, increased.

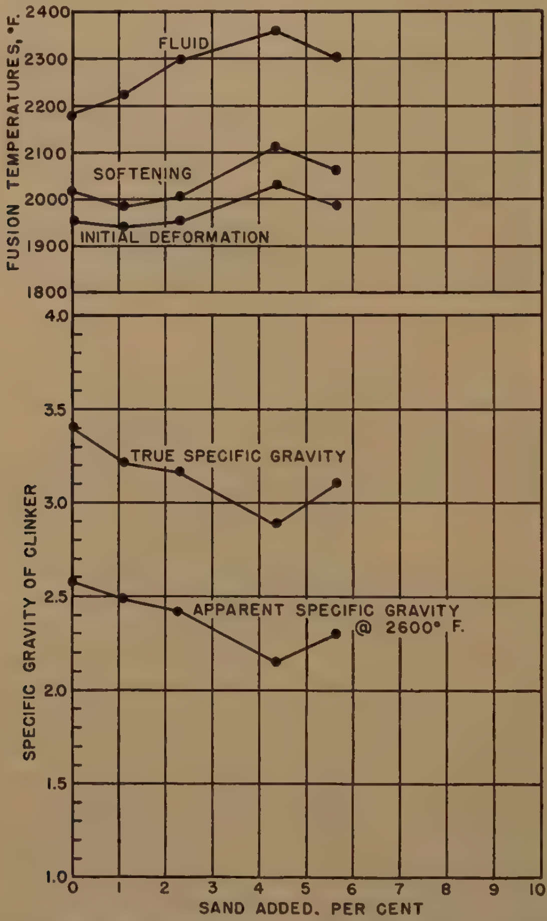


FIG. 9.—RELATION OF TRUE AND APPARENT SPECIFIC GRAVITIES OF CLINKERS AND OF FUSION TEMPERATURES OF ASH TO ADDITIONS OF SAND TO COAL.

The same lot of coal was used for a trial of the effect of additions of ash produced by low-temperature combustion of the coal. In this way, the content of the ash was increased without change in its composition. The results were as follows:

Ash added, per cent.......	0.0	1.1	2.3	4.3
Apparent sp. gr. at 2600°..	2.6	2.5	2.4	2.4

The effect of the additions of ash of the same composition are thus seen to be slight.

PRACTICAL CONTROL METHOD USED AT MINE

For practical use of the apparent specific gravity of the clinker as a criterion of the clinkering characteristics of the coal, and for modification of the apparent specific gravity by the addition of silica or other material to the coal at the mine, to suit the coal to the particular requirements of a customer, it is essential that some rapid method for the calculation of the amount of addition to be made shall be available. It is also desirable that the method be accurate to permit the addition of the minimum quantity of the ash-forming addition. Although the additional material serves a useful purpose in effecting more satisfactory performance of the coal, excessive additions are undesirable, as any addition reduces the calorific value of the coal.

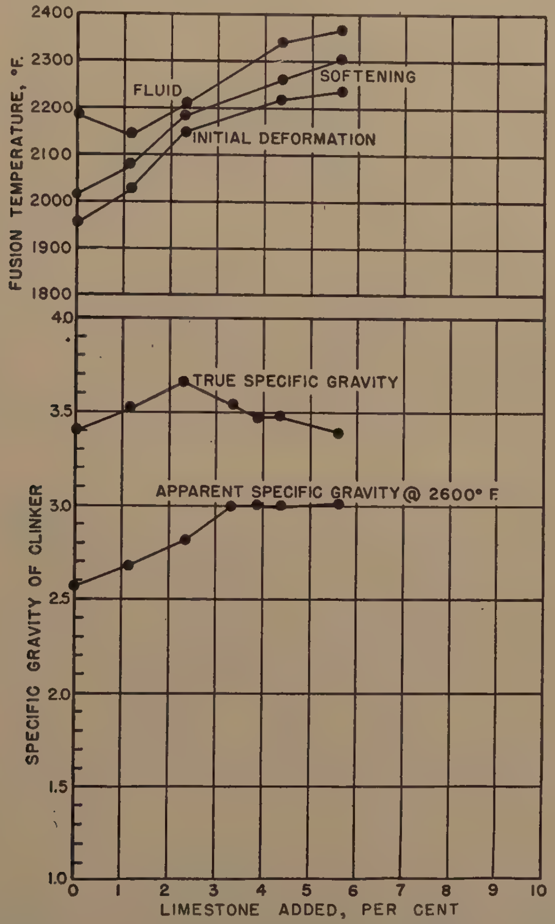


FIG. 10.—RELATION OF TRUE AND APPARENT SPECIFIC GRAVITIES OF CLINKERS AND OF FUSION TEMPERATURES OF ASH TO ADDITIONS OF LIMESTONE TO COAL.

Fig. 11 shows a type of chart that has been worked out from experience with the coals produced by the Northern Illinois Coal Corporation as burned in its customers' plants. Three regions are shown: the upper region is one of highly fluid ash, which forms flat clinkers that have a high specific gravity; the middle region is one of satisfactory clinkers which do not flow down upon the grate but move along with the fuel bed to be discharged uniformly; the bottom region is one of unfused or very viscous ash, which form voluminous masses of ash or clinker of low specific gravity.

Three boundaries of fuel-bed temperature are also shown. Taking, for example, the boundaries of the 2400° region: coals with 4 per cent ash content, whose clinker has a specific gravity of 2.0 to 2.6, will be satisfactory in furnaces where the fuel-bed temperature is of this order. As the ash content increases, it has been found that the allowable specific gravities increase to a range of approximately 2.4 to 3.0. For fuel-bed temperatures of the order of 2800°, corresponding to high-duty furnaces, the range of satisfactory clinker is 1.2 to 1.9 for a coal of 4 per cent ash and 1.6 to 2.3 for a coal of 12 per cent ash.

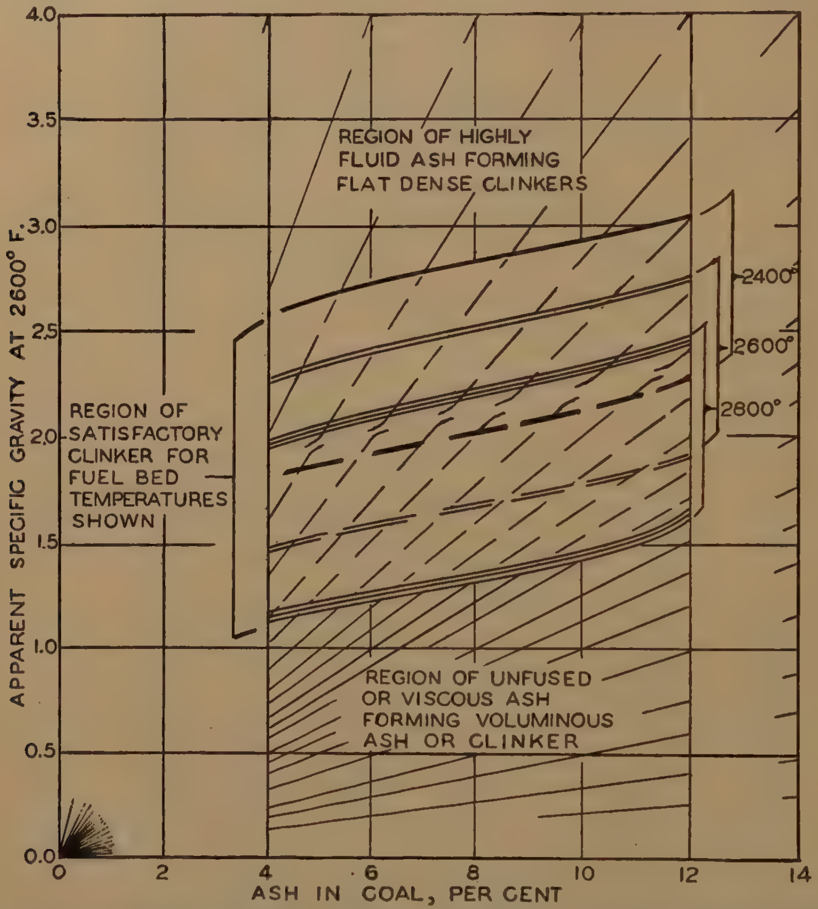
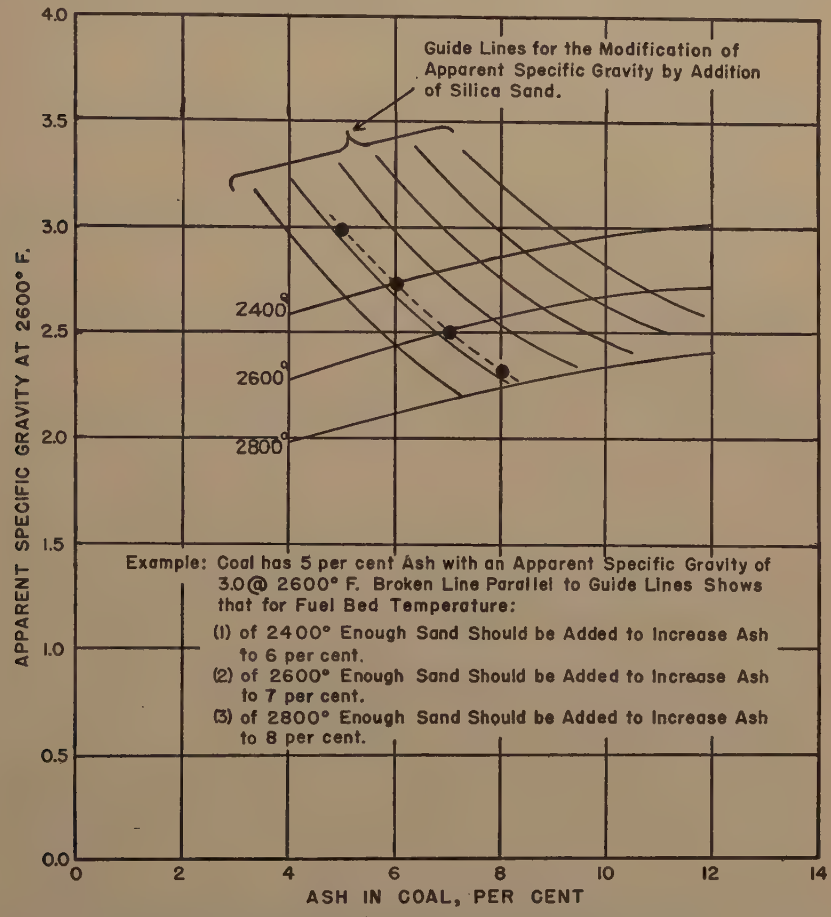


Fig. 11.—Approximate relation between ash content of coal, apparent specific gravity of clinker, and expected clinker difficulty for various fuel-bed temperatures.

Fig. 12 shows a working chart for calculation of the amount of silica to be added to coals of various original ash contents, to bring them into a range of satisfactory operation for use at various fuel-bed temperatures. The curves marked 2400°, 2600°, and 2800° have been found by experience, and are the upper limits of the temperature ranges shown in Fig. 11. The guide lines sloping downward from left to right are located on the basis of trials with the company's coals such as were shown in Fig. 9.



Guide Lines for the Modification of Apparent Specific Gravity by Addition of Silica Sand.

Example: Coal has 5 per cent Ash with an Apparent Specific Gravity of 3.0@ 2600° F. Broken Line Parallel to Guide Lines Shows that for Fuel Bed Temperature:

(1) of 2400° Enough Sand Should be Added to Increase Ash to 6 per cent.

(2) of 2600° Enough Sand Should be Added to Increase Ash to 7 per cent.

(3) of 2800° Enough Sand Should be Added to Increase Ash to 8 per cent.

FIG. 12.—TYPICAL WORKING CHART FOR CALCULATION OF AMOUNT OF SILICA SAND TO ADD TO COAL FOR MODIFICATION OF CLINKERING CHARACTERISTICS OF ASH.

An example of the use of the chart is shown in Fig. 12. A coal having an initial ash content of 5 per cent and an apparent specific gravity of the clinker at 2600° of 3.0 is to be modified to make it suitable for use in a low-duty furnace that has a fuel-bed temperature of 2400°. Locating the point 5.3 of the chart, a line is traced concentric with the guide lines to below the 2400° line. This coincides with an ash content of 6 per cent; the apparent specific gravity of the resultant clinker may be expected to be approximately 2.7.

For fuel-bed temperatures of 2600°, enough sand should be added to increase the ash content to 7 per cent, when the clinker will have an apparent specific gravity of approximately 2.4. For a high-duty furnace whose fuel-bed temperature is expected to be of the order of 2800°, enough sand should be added to bring the ash content to 8 per cent; the apparent specific gravity will then decrease to 2.2.

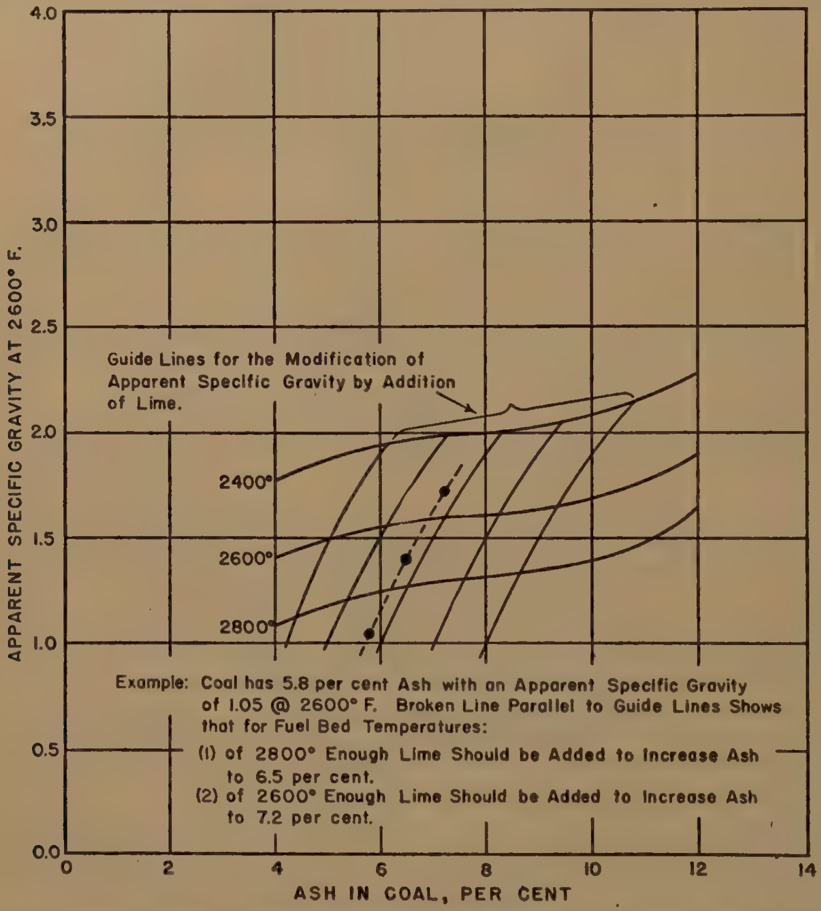


FIG. 13.—TYPICAL WORKING CHART FOR CALCULATION OF AMOUNT OF LIME TO ADD TO COAL FOR MODIFICATION OF CLINKERING CHARACTERISTICS OF ASH.

Fig. 13 shows a chart for use with coals whose clinkers normally have such a low specific gravity that they are too voluminous and retard the burning of the coal, and to which the addition of limestone will increase the specific gravity. The guide lines for control of the amount of the addition now slope upward from left to right; they are drawn from data such as are shown in Fig. 10. Examples of the use are given in Fig. 13.

The particular working charts shown in Figs. 12 and 13 are applicable only to the mines of this company. Others desiring to use this criterion and method of control would have to work out similar charts for their particular coals after tests with this method on the coals and with various addition agents.

Apparent Specific Gravity Contours of Mine

The use of charts such as those of Figs. 12 and 13 presupposes a knowledge of the ash content and apparent specific gravity of the clinker of the coal about to be treated and loaded in the tipple. Even though the

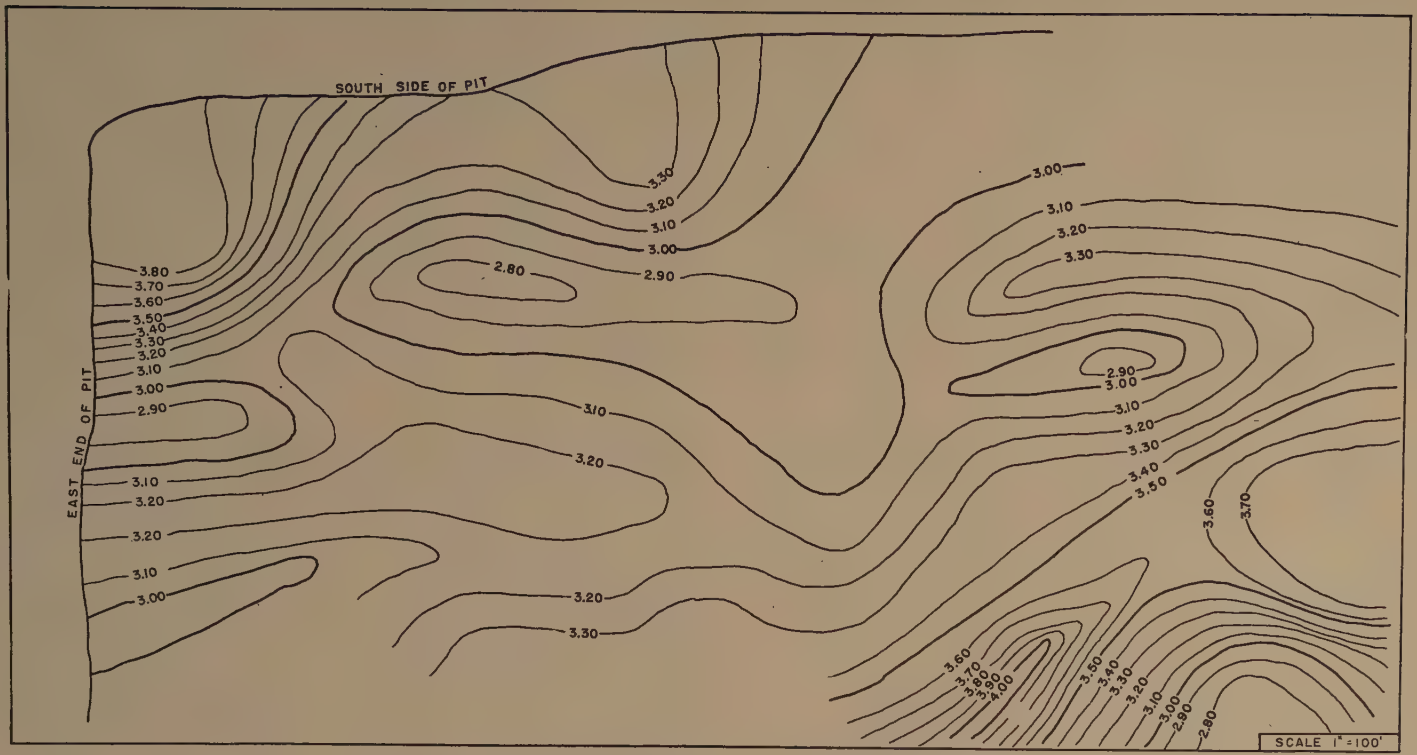


FIG. 14.—CONTOUR LINES OF CONSTANT APPARENT SPECIFIC GRAVITY.

method for the determination of the apparent specific gravity is relatively simple, it cannot be run while the coal is in the tipple. It is necessary, therefore, that the mine be surveyed beforehand.

Table 6.—*Comparison of Specific Gravities of Clinkers from Untreated and Treated Coal from Laboratory Tests and of Treated Coal and Clinkers from User's Plant*

As Mined		As Shipped			From User's Plant			Plant Clinker	
						Specific Gravity		Specific Gravity	
Ash, Dry Basis, Per Cent	Apparent Sp. Gr.[a]	SiO_2 Added, Per Cent	Ash, Dry Basis, Per Cent	Apparent Sp. Gr.[a]	Ash, Dry Basis, Per Cent	True[a]	Apparent[a]	Apparent[b]	True[b]
6.2	2.9	3.6	9.8	2.2	9.4	3.2	2.7	2.5	3.0
6.3	2.4	3.3	9.6	1.9	9.5	2.9	2.2	2.0	2.8
5.2	2.9	4.5	9.7	1.9				2.0	2.8
5.5	3.3	5.0	10.5	2.0				2.1	2.8
5.2	2.7	3.0	8.2	2.3	8.2	3.1	2.3	2.4	3.3
5.4	3.1	2.0	7.4		7.3	3.5	2.6	2.5	3.4
3.9	2.4	1.0	5.5	2.1				2.3	3.1
7.6	2.9	4.0[c]	11.2	2.1	11.2	3.2	2.2	2.4	3.2
6.0	2.9	4.0[c]	9.6	1.8	9.6	3.2	1.8	1.9	2.9
7.0	1.7	2.0[d]	8.2	2.1	8.2	3.1	2.1	1.6	

[a] From standard test-furnace burnings.
[b] Determined on clinker from user's plant.
[c] 4 per cent clay.
[d] 2 per cent limestone.

Fig. 14 shows contour lines of constant apparent specific gravity of the clinker at 2600° for a section of one of the working pits of the Wilmington mine of the Northern Illinois Coal Corporation. With this chart, charts such as Figs. 12 and 13, and a knowledge of the equipment in use in the customers' plants, the operator can load coal without treatment, or with the correct amount of the proper treatment to meet the required conditions of operation.

Practical Results of Use of New Criterion

The use of the criterion of the apparent specific gravity of the clinkers in control of the shipments from the mines of the company has proved highly satisfactory over the past several years. The best control of shipments is considered to be the selection of coal from the current production whose ash will have specific-gravity characteristics suited to the requirements of the customer's plant. Blending is used when possible, and the addition of silica, lime, or other materials when necessary.

Table 6 shows the results of the treatment of typical coals as revealed by a comparison of the ash contents and specific gravities of clinkers from the laboratory furnace before and after treatment and of clinkers from the treated coal as sampled at the user's plant and of clinkers from the user's furnace. Inspection of this table will show that the apparent specific gravity of the clinkers formed in the laboratory test furnace from coal sampled at the user's plant and that of clinkers taken from the user's furnace usually checked quite closely with that obtained on the sample taken at the mine. Close agreement is also shown between the true specific gravity of the clinkers from the test furnace and from the user's furnace.

A few examples of the effect of the additions on the performance of typical plants will be of interest. One customer operated both a slag-tap, pulverized-coal-fired furnace and underfeed stokers. The former requires a coal giving a highly fluid ash—that is, one with high apparent specific gravity; the stokers require a coal giving a clinker of low specific gravity. By proper selection of coals from the various sections of the mine, and by treating when required, the proper coals for the two applications are shipped to the customer. The cars are marked for the application to which they are to be put.

In a burning test of 10 cars of coal in a railroad locomotive, the coal failed to give satisfaction because the ash was of low apparent specific gravity. Both a coal specially selected because of its higher specific gravity and two cars of the first coal treated with additions of lime proved satisfactory. The heating value and ash content of all coals were similar.

Certain users have seasonal load demands in which extremely high loads are placed on the boiler plant for short periods. They had serious clinker trouble during these peak periods. By treatment of the coal to decrease the apparent specific gravity of the clinker, satisfactory operation was obtained during the peaks. After the peak had passed, coal having clinker of normal specific gravity was again shipped to the plants.

The equipment installed for the treatment of coal consists of a central supply bin, a conveying system to feeding bins, and vibrating feeders by which the admixture can be applied at uniform rates up to 5 per cent by weight of the coal with an accuracy of 0.25 per cent.

Over the last three-year period, the tonnage treated and the amounts of sand used have been as shown in Table 7.

TABLE 7.—*Tonnage Treated and Sand Used*

Year	Coal Treated, Tons	Percentage of Mine Output	Sand Used, Tons	Average Addition of Sand, Per Cent
1939	290,000	25	7,600	2.6
1940	230,000	18	6,000	2.5
1941	170,000	15	3,800	2.2

The decrease in the percentage of the coal treated, and of the amount of addition made, are the result of a better knowledge of the selection and treating from the accumulated experience and to a slight decrease in the apparent specific gravity of the clinker of the coal as mined.

Without a method for the control of the amount of additional material, virtually all of the coal would require the maximum treatment, whereas with the control the amount added varies from 0 to 5 per cent. A regular addition of 4 to 5 per cent to all of the coal mined would be prohibitive, because of the increase in the ash content of the coal with no offsetting beneficial result.

Some segregation of the admixture is known to occur but sand passing the 100-mesh sieve adheres fairly well to the coal if it is not subjected to excessive handling and drying. A local clay that adheres better than the sand is available, and a smaller amount is required to give comparable results, but the appearance on the coal is unsatisfactory.

SUMMARY

This paper has presented a new criterion for the clinkering characteristics of coal ash; namely, the apparent specific gravity of the clinker formed when the coal is burned under controlled conditions simulating those of industrial underfeed stokers. The method used in the burning tests requires no more coal than a gross sample taken for preparation of the usual laboratory sample and the total time and expense are little more than for the usual ash-fusion determination.

The apparent specific gravity of the clinker has been shown to increase with increase in the fuel-bed temperature, and a method for correction to a standard temperature of 2600° has been described.

The apparent specific gravity has been shown to be roughly related to the other measurable characteristics of coal ash, the chemical composition and the fusion temperatures, and has been shown to vary more regularly with additions of extraneous ash-forming material than do the fusion temperatures.

By laboratory test and by plant experience, the additions of silica, fire clay, and limestone have been shown to effect desirable changes in the clinkering characteristics of coal ash. Fire clay and silica decrease and limestone increases the clinkering.

A practical control method* has been described by which the amounts of admixtures required may be quickly calculated when the apparent specific gravity and ash content of the coal to be treated and the operating characteristics of the user's plant are shown. Although the working charts presented are not generally applicable to all coals, similar charts can be worked out for other mining properties, and it is the hope of the author that others will try out the new criterion and report the results that they obtain.

ACKNOWLEDGMENT

A development of this kind, requiring a constant search for the significant characteristics of coal ash and at the same time setting up a useful method based on the findings, must necessarily be the work of many heads and hands in an organization. Grateful acknowledgment is made for this assistance with particular reference to the work of R. M. Brawner, F. W. Uthoff, Melbourn McKee, and A. C. Guill. Thanks are also due to Ralph A. Sherman, supervisor, and Bertrand A. Landry, assistant supervisor, Fuels Division, Battelle Memorial Institute, for their helpful assistance in interpreting the results of the work.

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* Editor's Note: It is understood that a patent application covering this method of evaluating the clinkering characteristics of coal ash and its control was filed by the author before his death.

Practical Anthracite Combustion

By J. F. K. Brown,* Member A.I.M.E., and E. E. Roecker†

(New York Meeting, February 1942)

For three years The Hudson Coal Co. has used egg anthracite instead of coke in its foundry cupola. It has long passed the stage of being told it cannot be done—the metal would be cold, of poor quality, or the speed of operation would be low. From observation, from use and from test, we know that our metal is about average; the cast iron invokes no adverse comment from the machine shop and the speed suits our foundry requirements. We have therefore resolved into practice anthracite combustion processes in an industrial furnace formerly using coke, under conditions suitable to our needs.

When it comes to translating what we know into use in another cupola, however, where operating conditions may be different, or to explaining why, with seemingly the same conditions in our own cupola, we have attained sometimes a temperature of 2800° in the metal and at another time 2600°, or a speed of 5 tons per hour against 2½ tons per hour, we cannot do it. We cannot do this because there seems to be an unbridged gap between laboratory ideas on combustion and practical ideas on combustion, which we have been unable to cross in terms understandable to us as practical consumers. This gap or bridge centers around the problem of how combustion actually takes place; and we believe it is controlled by two factors: (1) the character of the surface of the fuel and (2) the space provided and needed within which the oxygen of the air is allowed to or can act on that surface.

It has long been contended on behalf of coke that porosity is an important factor in combustion, although, admittedly, there are two schools of opposing thought on this question, as revealed in technical literature. It has been understood generally that a light, porous coke is the best metallurgical fuel because there are more square inches of carbon exposed to the blast of air, therefore more useful carbon-dioxide reactions can take place in the given space between the entrance of the air and the point where useful work is done. It is now our general idea that the heavier and the harder the coal, the better the metallurgical fuel, provided the surface of that fuel is suitable for fast reaction with the oxygen.

Time and Space

A cupola is loaded in layers of fuel and iron. The horizontal line where the first charge of iron rests on the coal is the top of the bed coal. With us that is 22 in., usually, above the tuyeres where the air enters. This means 600 lb. of coal. Every subsequent coal layer is 100 to 125 lb. of coal, and that 100 to 125 lb. of coal melts 1000 lb. of iron at around 2700° in the ladle. This quantity of coal represents a 3-in. layer spread across the cupola. The question arises: If this 3 in. of coal will melt 1000 lb. of metal, why was it necessary to use 600 lb., or 22 in., of coal beneath and between the first charge of iron and the air entrance?

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Our first attempt to answer that question was to lower the bed level from 22 in. above the tuyeres, by stages, to 7 in. above the tuyeres. As we went down, however, the metal melted faster (which in itself was interesting) but the metal temperature decreased and finally could just about be poured into the molds. This meant that there was a time-space factor involved—viz., time for the oxygen to pick off the carbon from the coal within a space sufficient for that to be accomplished—whereby reasonable metal temperatures were attained, or, in other words, we had to have 22 in. or thereabouts above the tuyeres to the first layer of iron, irrespective of the fact that there appeared to be far more coal (B.t.u. available) than necessary to accomplish the result desired. This was our first inkling that design of apparatus for coke represented apparatus much too huge for anthracite.

Having found that this space between tuyeres and metal was necessary, we asked ourselves whether that space must be filled with coal. Anything else we might put within that space had to be a noncombustible and something that did not melt or affect the iron. The logical possibility was firebrick from the cupola lining. Therefore we took out 500 lb. of coal from the bed and replaced it with a mixture of 250 lb. of coal and 250 lb. of broken firebrick. This mixture was placed in the cupola in the same position as the bed coal, with the addition of 100 lb. of coal on the top to do the actual melting, thus making a total of 600 lb. of material in the bed, the same quantity as when coal alone is used, and thus maintaining the same space between tuyeres and metal. We ran 15 tests with coal and brick and compared them with 15 tests of coal only. The results are given in Table 1.

These tests indicated that with less coal but the same space we could get hotter metal with the coal and brick than we did with the coal alone. It seems to us, therefore, very important that the factors operating within this required space should be determined, because these are the same factors that control practical combustion, whether it is in a cupola furnace or an ordinary household furnace, and they all vitally affect design.

TABLE 1.—*Comparative Tests*

Tests with Coal Only						Tests with Coal and Brick						
No.	Volume, Lb.	Height of Bed, In.	Temperatures, Deg. F.			No.	Volume, Lb.	Weight of Brick, Lb.	Height of Bed, In.	Temperatures		
			1st Tap	High	Last Tap					1st Tap	High	Last Tap
112	1,888	24	2690	2720	2600	113	1,800	100	25	2720	2780	2700
103	2,090	26	2700	2720	2640	108	1,870	100	25	2660	2740	2720
102	2,290	26	2700	2720	2660	106	1,857	100	26	2700	2750	2670
101	2,142	23	2680	2680	2530	105	1,810	100	25	2670	2760	2750
97	1,811	23	2680	2730	2690	104	1,830	100	25	2660	2760	2760
95	1,900	24	2690	2730	2720	92	1,975	125	20	2650	2740	2700
91	1,743	20	2680	2720	2680	85	1,670	250	20	2660	2730	2700
88	1,658	20	2690	2740	2700	83	1,644	300	20	2600	2750	2720
69	1,420	22	2660	2720	2620	81	1,452	280	20	2650	2800	2750
68	1,455	20	2680	2740	2740	80	1,418	250	20	2640	2740	2730
67	1,441	18	2670	2720	2650	79	1,656	250	20	2680	2780	2720
66	1,524	18	2640	2740	2680	73	1,563	300	20	2600	2760	2760
65	1,554	20	2690	2740	2620	72	1,620	300	20	2600	2790	2790
62	1,994	20	2680	2720	2600	71	1,538	300	24	2620	2760	2720
61	1,623	20	2700	2740	2650	70	1,611	300	20	2600	2780	2760
Average			2683	2725	2652					2647	2761	2730

PREHEATED ANTHRACITE

About this time our attention was drawn to some Russian experiments on the use of carbonized anthracite, in which it was stated that by preheating the anthracite for a considerable number of hours the melting results were materially improved. We tried the same thing. We heated anthracite, egg size, for about 18 hr. in fashion similar to that followed in making beehive coke; that is, we heated it by its own combustion. The resultant material is dark, having lost its black shine, and on many pieces the outer skin has a slight coke luster. The coal is decidedly heavier, it sinks at the specific gravity at which the normally prepared coal would float, it absorbs water to some extent, the character of the ash is changed and it has porosity in the form of cracks and crevices, if porosity means anything. In effect, the anthracite had been "coked" on the outside and the physical changes described had occurred. We used this material in the cupola first as we used the ordinary coal, and once more the results were materially higher than those we obtained with the ordinary fuel. Instead of an average of 2700°, we found ourselves running close to 2800°, all other conditions being the same. This result checked with the stated results of the Russian experiments. Our results were as shown in Table 2.

TABLE 2.—*Effects of Heating Anthracite*

Number of Tap	Cupola Run Using Heated Anthracite for Fuel, Experiment No. 215			Cupola Run Using Raw Anthracite for Fuel, Experiment No. 212		
	Temperature of Metal, Deg. F.		Weight of Metal Tapped Out, Lb.	Temperature of Metal, Deg. F.		Weight of Metal Tapped Out, Lb.
	At Spout	Pouring		At Spout	Pouring	
1	2700		1,700	2680	2580	1,500
2	2750		900	2720	2620	1,500
3	2750		1,100	2740	2630	1,500
4	2770		1,400	2740	2630	1,400
5	2800		1,600	2730	2620	1,500
6	2800		1,600	2730	2610	1,500
7	2790		1,600	2740	2610	1,400
8	2820	2700	1,600	2740	2610	900
9	2800		1,600	2720	2620	800
10	2820	2720	1,600			
11	2790		800			
12	2790		500			
Total			16,000			12,000
Average	2782			2727		

Both runs were conducted under practically the same conditions, including air volume, pressure, etc. The run using the heated anthracite had a faster melting rate, and there was less clinkering in the bed coal.

Thus a point is added to our argument that the character of the surface of the fuel and the space in which the air has to operate are the two most important practical factors of anthracite combustion. Here was a heavier fuel, which did much better than the ordinary anthracite, and, since it *was* a heavier fuel, apparently the carbon in the surface of each lump had changed its character to the extent of influencing the rate at which the carbon passed to the oxygen.

MECHANICS OF COMBUSTION

The commonly accepted laboratory picture of the mechanics of combustion is that of a sphere of carbon surrounded by an unstable gas, which is continually in motion and through which passes sufficient carbon to unite with oxygen on the outside layer of this surrounding gas. But this theory does not go far enough in practical combustion. Are we to assume that these spheres of carbon part company from the larger pieces forming the regular coal bed, in sufficient number so that they more or less represent a flame phenomenon—thousands to millions of them moving upward through a given space in a given time until the reactions of the carbon through this sphere of gas attain their greatest maximum at the point where the melting takes place, which, in our case, is 22 in. above the entrance of the air? Or, on the other hand, are we to assume that this filament of gas surrounds each lump of the coal that is in the bed and that a similar diffusion of carbon takes place over the surface of the larger pieces? Under the first condition we should suppose that the carbon particles jump from the larger piece to the oxygen in the air stream and thereafter combustion, being the absorption of these particles in that air stream, is a rapidly moving action, continually forced upward. In the second case we have to suppose the oxygen going after the carbon on the face of each lump of coal and combustion as the absorption of the oxygen by the carbon at fixed locations. It is likewise conceivable that the location where the transposition of the heat generated into useful work, as determined by design of apparatus, whether that be a layer of iron or a water tube, could be very largely influenced by the size, shape, depth and width needed to allow either of these conditions to take place, and that the shape, depth, width, could be different for the one combustion conception over the other. Whatever may be the real answer, the outstanding feature to us is that for good combustion the surface of the carbon must part company with these spheres of carbon very rapidly, and hence the character of that surface in relation to this problem becomes, in our mind, extremely important.

We believe that the anthracite that does its useful work in the cupola, and in any other process, is a different anthracite from that to which the normal experimental and physical reactions by laboratory tests are attributed; that is, the glossy, black anthracite as it is prepared. We believe that anthracite as it passes down the cupola begins to lose its volatile matter; that the character of its ash changes; that the character of the carbon is altered; that its weight increases, so that when it gets down to the point where it is actively melting the metal, the original anthracite has been altered to something like the substance we obtained with our preheated or "coked" anthracite. This means that a great deal has still to be learned about practical combustion and, as stated before, we believe the problems center around what we have termed the time and space factor and the character of the carbon on the surface of the fuel. These combustion features, when resolved, will point to a possibility that anthracite-using apparatus could be built much smaller, involving much less capital expenditure, for the same result. This we believe to be the outstanding economy in the use of a dense, concentrated fuel. From experience we can put 4500 lb. of metal into our cupola when we use coke, while we can put 8000 lb. of metal in when we use anthracite. Thus our cupola is almost twice the size we need for our purpose when we use anthracite. We could have spent less money on equipment, for the same result.

Coke Formation in Domestic Stokers

By Charles H. Sawyer* and Walter Knox†

(New York Meeting, February 1944)

All of the coals commonly used in domestic bituminous stokers form coke, and satisfactory operation depends upon the fact that the coke formed is so weakly bound together that it breaks readily in the normal operation of the stoker and automatically maintains a good fuel bed.

Some coals, because of their tendency to form solid, blocky coke, are practically excluded from use in small domestic stokers as now built. Other coals, some among the best stoker coals available, can form this troublesome coke under certain conditions.

Many contradictory methods have been offered as cures for coke formation. Generally the procedure has been to try various adjustments until a combination was found that seemed to give better results. Such random methods of treatment have not provided any reliable information as to causes.

The present paper covers a comprehensive test program undertaken by the Koppers Company's Stoker Coal Research Laboratory at Kearny, N. J. The program had two main objectives; first, to discover what factors affected coke formation, and second, to define corrective steps that could be applied in stoker service work.

Conclusions and Results

1. The design of the stoker outweighs in importance all other factors capable of improving coke formation.

2. Insufficient air supply aggravates the formation of large blocky coke, although raising the air above 30 to 40 per cent excess (as determined during steady running) does not further improve the character of the coke formed and only reduces its accumulation at the expense of efficiency, increased fly ash, poorer clinkering, and other disadvantages.

3. Increasing the coal feed results in a somewhat more friable coke structure and is recommended for persistently troublesome jobs.

4. Increasing over-fire draft improves the character of the coke formed and reduces troublesome accumulations by accelerating off-period burning, although at some expense in efficiency and some increase in fly-ash accumulation.

5. Coke structure was not substantially improved by blocking some of the retort tuyeres, although the blocking of tuyeres at the feed-screw end of a rectangular retort did lead to somewhat improved performance, possibly because of considerably better distribution of coke in the fuel bed.

6. With the equipment tested no difference was shown between installation of the same stoker in a steam boiler and a warm-air furnace, or between a sand-filled and a ventilated type of hearth.

7. The following design features had no noticeable effect on coke formation:

a. Presence or absence of coal-tube air as provided by many manufacturers to eliminate smoke-back.

b. Presence or absence of a flap damper in the air duct to minimize off-period burning.

c. Continuous or intermittent (ratchet and pawl driven) movement of fuel feed screw.

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† In charge of stoker research for the Koppers Company during most of the work covered by this paper. At present with the H. D. Conkey Co., Mendota, Illinois.

8. Reducing the size of a rectangular retort by blocking off one end to form a small square retort resulted in better coke formation when the proportion of "on" time was comparatively great, but showed little difference when operation was at lower demand rates (e.g., 7½ min. on and 52½ min. off).

9. Changes in the feed-screw mechanism which greatly reduced packing and crushing of the coal being fed resulted in a definite improvement in coke formation.

10. Although exact features of retort design that promote good coke formation were not established, coal turn, flare, and possibly shape and weight were shown to be characteristics deserving further investigation.

Scope of Work

Work on the effects of such factors as air supply, feed rate, draft, and tuyere plugging was given major consideration because of its obvious importance to the servicing of present installations.

However, as the tests progressed it became clear that something more than control of adjustable factors was involved in the problem of coke formation. Therefore the work was extended to include the other investigations indicated; viz. installation differences and principles of design.

Equipment and Methods of Test

The stoker laboratory has been described in detail in an earlier paper presented before this institute.[1] It consists essentially of two typical household units, one steam and one warm air, both so instrumented that data of interest in evaluating performance can be automatically recorded with a minimum of attention.

In the tests covered by this paper two different stokers were used interchangeably in the two heating units. One of these stokers (hereafter designated as stoker No. 1) produced a desirable type of coke formation, no matter what was done to adjustments of air, draft or coal feed; and the other (called stoker No. 2 in this paper) consistently made solid, blocky coke, which was only moderately improved by the best combination of adjustments.

It should not be inferred that stoker No. 2 is typical of its manufacturer's line. It is no longer manufactured, but was used for these tests because it embodied a number of principles that had been suspected of contributing to bad coke formation.

For the purpose of this investigation, good coke formation has been defined as that permitting the maintenance of an efficient and responsive fuel bed without a large enough accumulation of coke to make clinker removal difficult.

On that basis, a solid, blocky coke, which forms in long continuous trees, is bad, and a friable coke, which forms no trees or at least trees that break apart of their own weight, is good.

It is difficult to show differences in coke structure quantitatively. In this work all of the coke remaining in the fuel bed at the end of each test was recovered and weighed, and a screen analysis was made to show proportions of the following sizes; plus 4 in., 4 to 2 in., 2 to ½ in., and ½ in. to 0. Coke from each test was handled in exactly the same way, so that the size relations shown indicate quite fairly whether coke was solid or friable.

For the tests of which motion pictures could be obtained the following system of comparison was employed. After reviewing the films from all tests, an arbitrary five-point scale was agreed upon for coke character, in which the number one should represent the best coke obtained in the series and the number 5 the worst. A similar scale was arranged for accumulation, in which one should represent the slightest accumulation observed and 5 the greatest. These constitute the numbers given under the heading "Movie Record" in the data. As recorded, the first figure represents quality, the second accumulation. Thus 4 to 3 refers to coke that is appreciably more solid than the average, which has accumulated to a fuel bed of about an average size (approximately 10 in. deep). The figures used are the averages of independent ratings made by two operators, who viewed the movies shown out of order and unidentified as far as possible. These data agree remarkably well with the physical test data on coke residues, in most instances.

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[1] References are at the end of the paper.

The long-standing confusion regarding the problem of coke formation suggested that it was probably influenced by a number of factors, possibly interrelated. Because of this a number of tests were run, but in each as far as possible only one factor at a time was varied.

Throughout the test program one coal was used—a commercially prominent, washed, oil-treated, ¼ to ¾-in., Pocahontas No. 3 from McDowell County, West Virginia. A typical laboratory report on this coal is given in Table 1.

TABLE 1.—*Typical Analysis of Coal*

	PER CENT
PROXIMATE ANALYSIS	
Volatile matter	16.8
Fixed carbon	77.3
Ash	5.9
Sulphur	0.62
ULTIMATE ANALYSIS	
Carbon	85.92
Hydrogen	4.33
Oxygen	2.10
Sulphur	0.62
Nitrogen	1.14
Ash	5.89
SCREEN TEST	
On ¾ in. rd	0.73
On ½ in. rd	26.13
On ⅜ in. rd	25.01
On ¼ in. sq	30.92
On 4 mesh	2.66
On 8 mesh	6.92
On 28 mesh	4.64
Through 28 mesh	3.62
Weight per cubic foot	45.3 lb.
British swelling Test No.	4½
B.t.u. per lb. (dry)	14,860
Fusing point of ash	2295°F.

Although only one coal was used in these tests, undoubtedly a considerable portion of the findings will be found to be applicable not only to coals of the same general character but to other widely used stoker coals as well.

Except as otherwise noted, each test included six days of varied operation, thus permitting one test to be concluded weekly, with a day allowed for sampling residues. The following schedule for operating changes was employed:

Tues. 9:30 a.m. to 1:30 p.m.	Fire lighted and stoker operated continuously.
Tues. 1:30 p.m. to Wed. 4:30 p.m.	7½ min. on and 7½ min. off. (Part A considered to start at 4:30 p.m. Tues.)
Wed. 4:30 p.m. to Thurs. 4:30 p.m.	7½ min. on and 22½ min. off. (Part B)
Thurs. 4:30 p.m. to Fri. 4:30 p.m.	7½ min. on and 52½ min. off. (Part C)
Fri. 4:30 p.m. to Sat. 9:30 a.m.	3 min. on and 57 min. off. (Part D, hold-fire).
Sat. 9:30 a.m. to Sat. 11:00 a.m.	Continuous operation (Part E, pick-up)
Sat. 11:00 a.m. to Mon. 9:30 a.m.	3 min. on and 57 min. off. (Part G, hold-fire)

This schedule stresses operation for comparatively short intervals, followed by relatively long "off" periods, and hold-fire operation. Such operation has been found by several investigators and by field experience to be particularly troublesome from the standpoint of coke formation.

TEST RESULTS

Effects of Air Adjustment

When a stoker is discovered making bad coke, probably the thing most easily and most often changed by the service man or home owner is the air adjustment of the stoker.

A considerable number of tests were run in which the air was varied. Significant data from one such series are shown in Table 2. Increases in the air supply resulted in progressive improvement in coke formation, as indicated by the quantity and size of

TABLE 2.—*Effects of Varying Air Supply*

NO. 2 STOKER, STEAM BOILER, COAL FEED 11 LB. PER HOUR, DRAFT 0.02 INCHES

Item		Test No.			
		IC-2[a]	IC-3	IC-4	IC-7
Air supply, per cent of theoretical		66.0	86.5	119.3	140.0
Pounds water evaporated per lb. coal fed	Part A[b]		6.18	8.30	7.91
	Part B		10.60	8.87	5.75
	Part C		7.00	5.15	3.59
	Avg.		6.71	7.75	5.92
Pounds water evaporated during 1½ hr. pickup			35.4	65.0	89.0
Total coal fed, lb		212.96	357.74	328.71	354.67
Average maximum stack temperature, deg. F.	Part A	370	430	554	608
	Part B		414	493	514
	Part C		408	409	395
Average retort temperature, deg. F.	Part A	145	190	190	205
	Part B		171	204	178
	Part C		185	214	173
Observed excess air[c]	Part A	41	21.5	46	90
	Part B		33	123	208
	Part C		164	308	425
Clinker, lb			2.40	4.38	3.35
Fly-ash, lb			0.58	0.30	0.74
Loose ash, lb			15.50	11.29	13.94
Coke residue, lb	+4 in.	2.56	7.53	1.84	none
	4 to 2 in.	11.67	7.32	9.39	7.10
	2 to ½ in.	21.07	8.42	9.00	8.92
	½ in. to 0		25.38	3.63	4.96
	Total		48.65	23.86	20.98
Movie record, see text for explanation	Part A	5–5	5–5	4–3½	3–3
	Part B		5–4	4–3	3–3
	Part C		5–4	4–3	4–3½
	Part E		poor	fair	fair
	Part G		5–4	4–3	4–3½

[a] In test IC-2 coke filled combustion space completely and caused a pin to shear during the first night of operation.

[b] Parts A, B, C, etc. refer to the standard schedule of operation described in the text.

[c] Observed excess air was calculated from the average analysis of spot samples of flue gas taken at the middle of "on" periods.

coke remaining after a week's test, and by the motion picture record of these tests. It is apparent that the last increase in air supply resulted in improvement only at the higher rates of operation.

Although tests on the effect of air setting definitely point to the importance of sufficient air supply, they show also that air far in excess of previously recommended values does not contribute materially to better coke formation. It is, however, worth adding that nothing observed during the tests supports the quite prevalent belief that too much air is as bad as not enough as far as coke trouble is concerned. Apparently the worst results of too much air are loss of efficiency and increased fly-ash; the coke does not get any worse, it simply fails to get any better, as air is increased beyond an optimum value.

Effects of Coal-feed Changes

To determine what effect coal-feed changes might have on coke formation, several tests were run in which coal feed was doubled.

Table 3 shows the results of two pairs of tests in which feeds of approximately 11 lb. per hour and 22 lb. per hour are compared. In both cases "on" periods were halved when the feed was doubled, so that the same amount of coal was burned during equivalent periods of the tests.

In the first pair (IC-4 vs. IC-8) air supply was relatively lower for the test at 22 lb. per hour. In spite of this, a considerable improvement in coke formation is shown by the size and quantity of coke residue and by the data from the motion picture record. Still greater improvement is shown for test

No. IC-32 as compared with No. IC-31, where the ratio of air supply to coal was more nearly equal.

The very much smaller size of coke in a test made at 22 lb. per hour feed as compared with 11 lb. per hour is well shown by the photographs obtained during a third series of tests, which were made in the warm-air furnace (Figs. 1 and 2).

Data from all these tests show some decrease in efficiency and considerable increase in fly-ash accumulation at the high feeds. These disadvantages might be considered insignificant as compared with the improvement obtainable in a bad coking stoker.

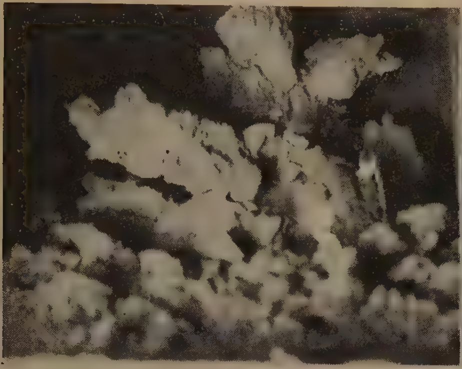

FIG. 1.—COKE STRUCTURE IN A TEST MADE AT 22 POUNDS PER HOUR COAL FEED.

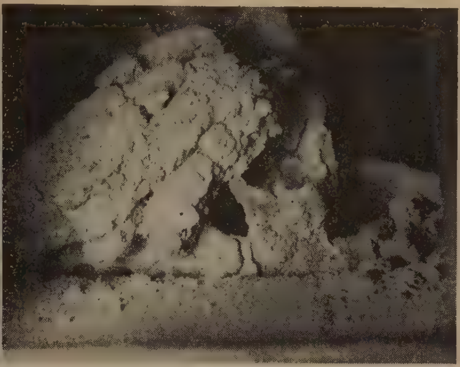

FIG. 2.—COKE STRUCTURE IN A TEST MADE AT 11 POUNDS PER HOUR COAL FEED.
Other conditions same as in test represented by Fig. 1.

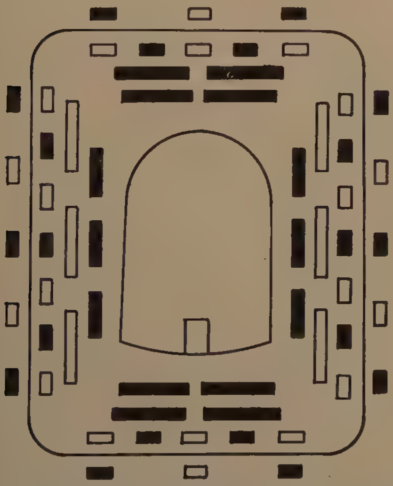

FIG. 3.—TUYERES SHOWN BLOCKED IN WERE PLUGGED WITH FURNACE CEMENT FOR MOST OF THE TESTS.

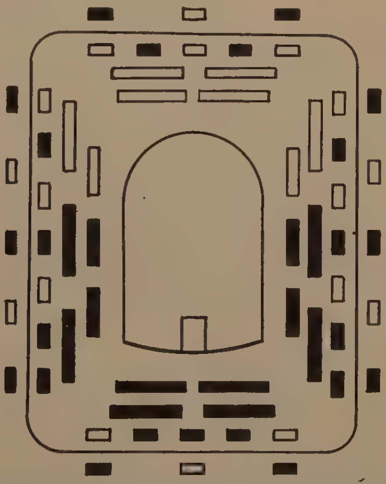

FIG. 4.—WHEN TUYERES SHOWN BLOCKED IN WERE PLUGGED WITH FURNACE CEMENT COKE FORMATION WAS SLIGHTLY IMPROVED AND DISTRIBUTION IN FUEL BED MUCH BETTER.

Effects of Draft

In the series of tests described in the Koppers Coal Company's booklet No. 1, "Effect of Stack Damper Setting or Furnace Draft on Stoker Performance," it was shown that increasing the draft decreased the fuel bed; that is, resulted in less accu-

TABLE 3.—*Effects of Changing Coal Feed*[a]
"ON" TIME HALVED AT HIGHER FEED RATES

Item		Test No.			
		IC-4	IC-8	IC-31	IC-32
Coal feed, lb. per hr		11.8	23.8	10.8	22.2
Air supply, per cent theoretical		119.3	93.0	118	117
Pounds water evaporated per lb. coal fed	Part A	8.30	8.93	8.32	7.57
	Part B	8.87	7.07	7.58	6.03
	Part C	5.15	4.22	4.57	2.95
	Avg.	7.75	6.44	7.00	5.86
Pounds water evaporated during 1½ hr. pickup		65	149	112	22.4
Total coal fed, lb.		328.7	426.6	327.1	344.0
Average maximum stack temperature, deg. F.	Part A	554		536	699
	Part B	493		460	627
	Part C	409		404	491
Average retort temperature, deg. F.	Part A	190		221	243
	Part B	204		195	244
	Part C	214		239	239
Observed excess air, per cent	Part A	46	66	68	161
	Part B	123	213	179	242
	Part C	308	581	311	583
Clinker, lb.		4.38	4.53	3.20	5.43
Fly-ash, lb.		0.30	1.25	0.55	1.27
Loose ash, lb.		11.29	16.12	13.90	12.05
Coke residue, lb	+4 in.	1.84	none	3.88	none
	4 to 2 in.	9.39	3.49	5.29	1.21
	2 to ½ in.	9.00	7.66	10.75	5.77
	½ in. to 0	3.63	5.59	10.52	5.08
	Total	23.86	16.74	30.44	12.06
Movie record	Part A	4–3½	2½–3	3½–4	2–2
	Part B	4–3	3½–3	3½–3	2½–2½
	Part C	4–3	4–3	4–3½	3–2½
	Part E	Fair	Good	Poor	Good
	Part G	4–3		4–3½	3½–2½

[a] Notes under Table 2 apply here.

mulation of coke. In order to find out whether this effect would be of any value with a stoker making really troublesome coke, a test was run at 0.07-in. draft in the No. 2 stoker, duplicating in all respects save draft a previous test, which had been run at 0.02-in. draft.

Data from these two tests are given in Table 4. Notice the very great difference in coke residues as shown by weights at the end of tests and by the movie record. There is obvious improvement in the nature of the coke formed but of course higher draft spells higher fly-ash and lower efficiency. In this particular pair of tests, the effects of these disadvantages were moderate, and it seems quite likely that increasing the draft might be worth while in troublesome cases.

Effects of Tuyere Plugging

As a result of a general investigation of stoker operation, C. A. Barnes concluded that, "an optimum tuyere area is indicated for a given fuel and desired range of operating conditions."[2] Certainly it seems reasonable enough that a retort designed for a given coal feed would have more tuyere area than would be needed for half that feed. On this assumption most of the tests in this series on coke formation were run after plugging about half of the tuyeres of the rectangular retort of stoker No. 2. Arrangement of the plugging was as shown in Fig. 3. However, a later test made with none of the tuyeres plugged indicated that actually there had been no advantage in the plugging as shown, possibly because in normal operation this stoker plugged about half its own tuyere area with fines.

Although several tests were run in which tuyeres were plugged in various ways, the only form of plugging that was shown to be of benefit was that represented by Fig. 4. Coke formation was slightly improved and distribution in the fuel bed was greatly improved by plugging the tuyeres in this fashion. The trend among stoker manufacturers toward reduction of tuyere area on the feed-screw side of retorts is supported by the results of this test.

TABLE 4.—*Effects of Changing Draft*[a]

Item		Test No.	
		IC-9	IC-10
Average over-fire draft, in. water		0.017	0.070
Coal fed, lb. per hr		12.1	11.9
Air supply, per cent theoretical		82	84
Pounds water evaporated per lb. coal fed	Part A	8.12	8.23
	Part B	9.18	7.83
	Part C	6.31	4.86
	Avg.	7.42	6.83
Pounds water evaporated during 1½ hr. pickup		65.2	51.7
Total coal fed, lb.		368.77	357.43
Average maximum stack temperature, deg. F.	Part A	543	549
	Part B	471	481
	Part C	413	377
Average retort temperature, deg. F.	Part A	278	225
	Part B	223	196
	Part C	219	166
	Part G	294	230
Observed excess air, per cent	Part A	25	102
	Part B	87	214
	Part C	271	459
Clinker, lb.		3.36	3.63
Fly-ash, lb.		0.39	0.53
Loose ash, lb.		13.78	14.43
Coke residue, lb.	+4 in.	4.75	none
	4 to 2 in.	8.19	4.39
	2 to ½ in.	10.33	6.53
	1.2 in. to 0	5.28	4.62
	Total	28.55	15.54
Movie record	Part A	4–4	2½–3
	Part B	4–3½	2–2½
	Part C	4–3	3–3
	Part E	fair	fair
	Part G	4–3	2½–2½

[a] Notes under Table 2 apply here.

In one pair of tests, with a smaller round retort which had been adapted to stoker No. 2, results were definitely better with all tuyeres open than with approximately half of the tuyeres blocked off.

In a pair of tests made with a small square retort, in which first only the top row of the original three rows of tuyeres was used, then the bottom row, examination of the semiplasticized ends of coke plugs removed from the retort indicated that the bottom of the plastic zone of the incoming coal was nearly flat, and that it lay approximately one-half inch below the open tuyeres in both cases.

This is somewhat different from the plastic zones indicated in fuel-bed pictures presented by Sherman and Kaiser,[3] where the bottom of the plastic zone assumes a cupped shape with edges at about the lowest tuyere level. The fact that operation in the present tests was characterized by relatively long "off" periods probably explains this difference.

In any case, plugging the bottom tuyeres of a stoker should raise the burning level and also the level at which incoming coal is being coked.

In general the effect of tuyere plugging on coke formation seemed to be less than has popularly been supposed.

Inadequacy of Controllable Factors

At the conclusion of the work described on variable or controllable factors, the problem appeared in the following light:

Stoker No. 1 made good coke, no matter what was done to air, draft or coal feed; and stoker No. 2, although some improvement was achieved, made bad coke in spite of the best possible combination of such adjustments.

Obviously something in the design or installation of these two stokers was responsible for this wide difference in coking behavior. The work on design and installation factors was limited in scope, but it is felt that some of the things shown by this part of the investigation are of great importance.

Installation Factors

Two installation factors were investigated. First, the two stokers used were interchanged from a steam boiler to a warm-air furnace and vice versa. No difference in the burning characteristics or in the coking behavior of either stoker was noticed.

Second, the original sand filling was removed from beneath the hearth of stoker No. 2 when it was installed in the warm-air furnace. Although the retort temperature of the stoker was then lowered appreciably, there was no difference in coking behavior. Considerable interest attached to this test on ventilated vs. sand-filled hearth because the ventilated hearth has been advanced so often as a cure for troublesome coke formation.

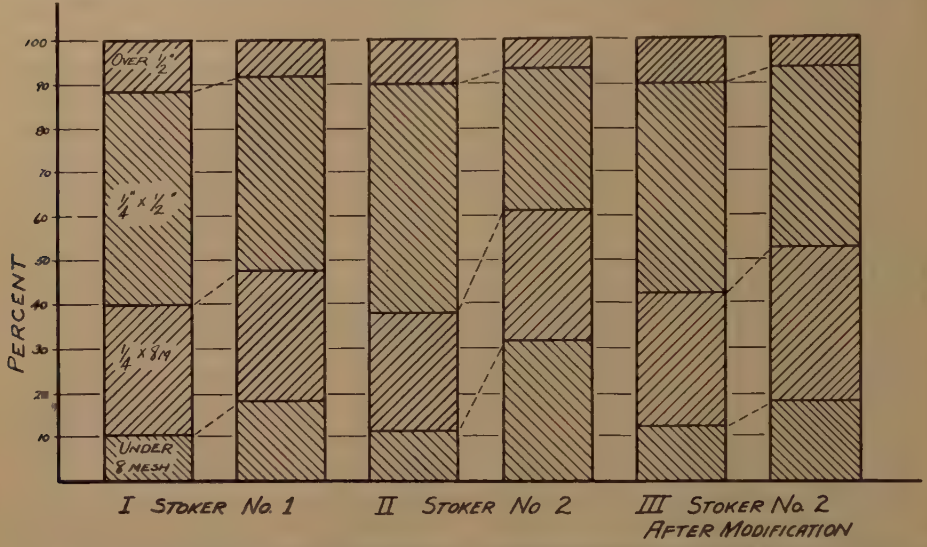

FIG. 5.—DIAGRAMS I AND II SHOW DIFFERENCE IN CRUSHING BETWEEN STOKERS 1 AND 2. DIAGRAM III SHOWS IMPROVEMENT OBTAINED BY MODIFYING FEED ON STOKER 2.

In each diagram, left column shows size distribution in original sample, right column distribution after feeding through stoker.

Design Factors

The general plan of the work on design factors was to eliminate successively the differences in design between the two stokers by altering one or the other. After each change a short burning test of the altered stoker was made to see whether there had been any important difference in coking behavior. In stoker No. 2 the changes that seemed promising were maintained, so that the effects might be cumulative, because the great difference in operation between the two machines was thought to be due to the sum of several factors of better design in stoker No. 1.

Three rather obvious differences in design can be disposed of rather shortly, since they were not shown to have any effect on coking:

First, neither blocking the air connection between air tube and coal tube in stoker No. 1, nor adding a cross connection from wind box to coal tube in stoker No. 2 (which originally had no provision for smoke-back air), had any noticeable effect on the coke formation.

It should be noted that Wright and Spicer[4] have reported that air admitted below the regular tuyere level was effective in improving coke structure only when used in conjunction with a vertical screw; which, of course, provides agitation not present in the conventional type of stoker here considered.

Second, the addition of a tight-fitting flap damper, positively closed by a damper motor during "off" periods, had no noticeable effect on the coke formation.

Third, a continuous feed mechanism was attached externally to the gear case of stoker No. 2, whose regular intermittent ratchet feed was then of course placed in neutral position. No difference in coking resulted.

One of the greatest differences between the two stokers was in the degree of packing in the coal tube. The feed screw in stoker No. 1 was smaller in diameter in the hopper than in the coal tube; the screw in stoker No. 2 was of constant diameter and pitch and in addition had a funnel-shaped opening at the spike trap, which caused coal to pack very tightly into the coal tube. In fact, packing was depended upon in this stoker to prevent smoke-back. The increased packing caused much greater crushing of coal in stoker No. 2 than in No. 1, as shown by diagrams 1 and 2 in Fig. 5. Sherman, Kaiser, and Limbacher[5] have shown that excessive fines tend to increase coke-tree formation. It was thought that relieving the packing in the screw of stoker No. 2 might improve coke formation by increasing the size of coal delivered to the retort.

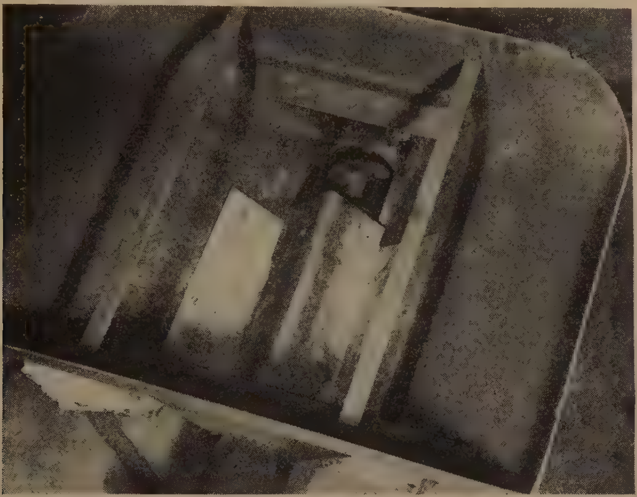

FIG. 6.—SHOWING TIN HOOD AND HALF PIPE COVER AT SPIKE TRAP ENTRANCE USED TO REDUCE PACKING AND CRUSHING IN STOKER NO. 2.

A photograph of the arrangement used to achieve this is shown in Fig. 6. All but two flights of the hopper portion of the feed screw were covered by a tin hood and the spike trap was replaced by a close fitting half-pipe placed over the screw to prevent the entrance of excess coal. Diagram 3 of Fig. 5 shows that crushing was thus reduced to compare very favorably with that encountered in stoker No. 1.

Coke formation was definitely better after this change but was still in no way comparable to that of stoker No. 1.

It had been noticed that the coal always pushed to the end of the rectangular retort opposite the feed screw of stoker No. 2, where it plugged the tuyeres, and that the coke trees in this stoker always arose from that end. The coal tube of stoker No. 1 had a sharper turn into the round retort and here coal distribution seemed better. In an attempt to improve distribution in stoker No. 2, a plaster distributing ledge was built into the bottom of the retort (Fig. 7).

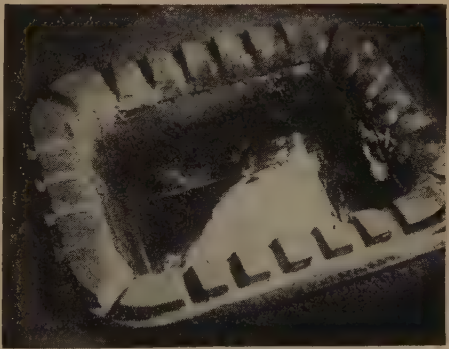

FIG. 7.—PLASTER DISTRIBUTING LEDGE ON RECTANGULAR RETORT OF STOKER NO. 2.

This ledge considerably improved distribution of coke in the fuel bed, and although there was no noticeable change in the nature of the coke formed, the better distribution led to more satisfactory operation.

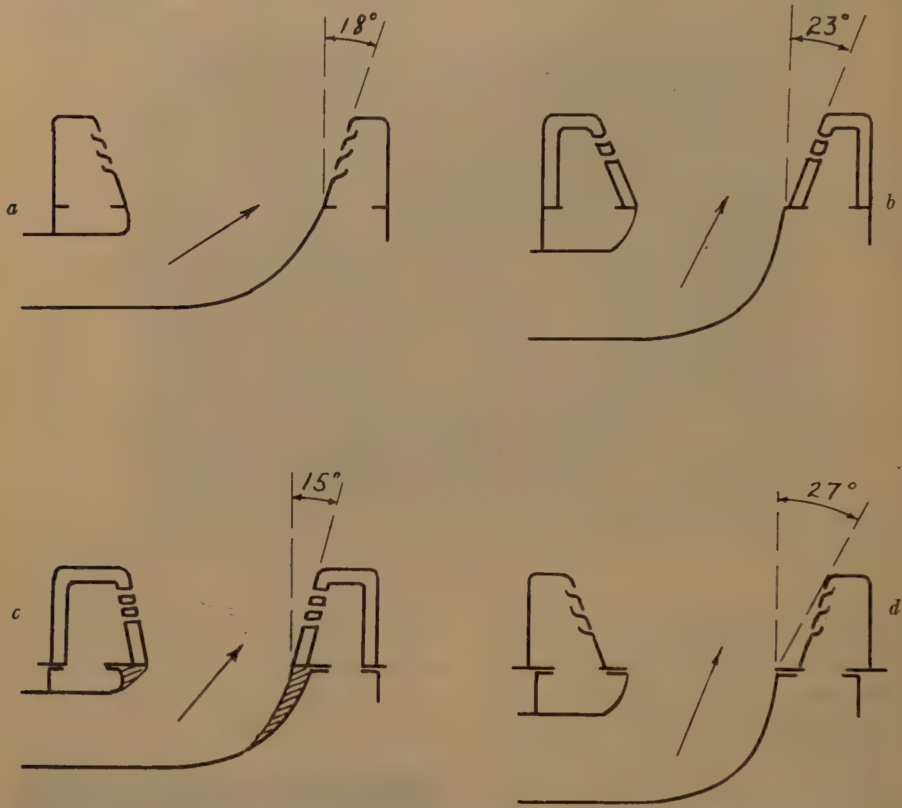

FIG. 8.—CROSS-SECTION SKETCHES OF FOUR RETORT STOKER COMBINATIONS.

Flare angles shown apply to all inside faces except in *d*, where sides have original flare of 18°. Arrows show approximate direction of coal entrance.

a. Stoker No. 2. Light-weight, sectional, rectangular retort, poor coal entrance (very poor coke).

b. Stoker No. 1. Heavy, solid cast, round retort, good coal entrance (very good coke).

c. Heavy, solid cast, round retort on stoker No. 2, fair coal entrance (good coke).

d. Light-weight, sectional, rectangular retort on stoker No. 1, good coal entrance (very good coke).

To see whether reducing the size of the retort might have the same effect as increasing the coal feed, a refractory false end was molded into the rectangular retort, to make a much smaller square retort. Tuyeres were cored through the false end. Later a steel plate was substituted for the refractory material, since it was thought the insulating effect of the latter might be undesirable. There seemed to be no difference in burning characteristics in the two methods. Both considerably improved coke formation at the higher rate of operation (7½ min. on and 7½ min. off) but were disappointing at 7½ min. on and 52½ min. off.

Retort Design

Although the coal-tube bend is really a part of the wind-box assembly, it and the retort are here considered as a unit, since a significant change in the bend was made by the adaptation of a round retort to stoker No. 2.

The retorts of stokers No. 1 and No. 2 differed widely. Stoker No. 1 (Fig. 8b) had a heavy, one-piece, cast round retort, with two rows of inside tuyeres. Its flare (the angle of its inside face with the vertical) was 23°. The coal turn was sharp, so that there was almost no tendency for the feed screw to push coal against the opposite retort face and to plug tuyeres on that side. It made very good coke.

Stoker No. 2 (Fig. 8*a*) had a much lighter, rectangular retort formed by three segments seating one on another within a large shell casting. Three rows of tuyeres were thus formed. Its flare was 18° and its coal turn was very shallow, permitting a considerable push against the face opposite the feed screw, with consequent plugging of tuyeres. It made very bad coke.

In order to determine whether the retort alone might be responsible for much of the difference in coking behavior, a round retort similar to that of stoker No. 1, except for less flare (15°) and one additional row of tuyeres, was adapted to the windbox of stoker No. 2. (Fig. 8*c*.) Since this retort overhung the original coal entrance, the coal turn was smoothed with plaster of Paris, as shown by the shading in the diagram. The coal turn was thereby altered to provide more nearly vertical entrance of the coal. The coke formation was good, almost as good as that of stoker No. 1.

Finally, the light-weight, rectangular, sectional retort of stoker No. 2 was adapted to stoker No. 1. (Fig. 8*d*) Since the retort was now longer than the coal entrance, a small ledge about ¾ in. wide was formed at either end, increasing the effective flare at the ends to about 27°. Flare at the sides remained 18°. The coal turn, of course, was the same as in the unmodified stoker No. 1, giving nearly vertical delivery with no tendency toward jamming against the end and plugging tuyeres. Coke formation again was very good, fully equal to that of stoker No. 1.

From a consideration of the four combinations described, it appears that the coal turn and the flare angle of the retort are two of the most important characteristics from the standpoint of coke formation. This conclusion is weakened by the observed behavior of the combination shown in diagram *C* where good coke formation was obtained in spite of the least flare of any retort used, but beneficial effects of the heavy solid casting of this retort may have more than offset this difference. Certainly all of these possible factors—coal turn, flare angle, weight of retort casting, and perhaps the shape (whether round or rectangular)—deserve further study.

Because the aims of the laboratory were to stress performance with various coals and various methods of operation rather than to study design, the subject was not pressed further. It is hoped that manufacturers will be encouraged to continue this work on design and that some of the things noted here may suggest methods of approach to the subject.

ACKNOWLEDGMENTS

Grateful acknowledgment is hereby extended to Mr. H. C. Waechter and Mr. W. H. Andren, who assisted in much of the test work, and to Mr. J. D. Doherty, who has lent invaluable assistance in the preparation of this paper.

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DISCUSSION

(Ralph Sherman presiding)

C. C. Wright,* State College, Pa.—I want to congratulate the authors on attacking this difficult problem. It is one that for a number of years has bothered most of us who work with domestic stokers. Several very interesting points have been brought out. I think flare angle of the retort, the angle of the tuyeres, and the design of the tuyere openings themselves are all very important. I am a little disappointed that the authors did not get around to studying the influence of the shape of the tuyere opening, as we believe this factor to be of considerable importance. With one or two stokers, we have found it necessary when burning slack coal to block off tuyere openings, while double screen coal from the same source did not require this change, at least not to the same extent.

In presenting the paper, Mr. Sawyer said that the addition of air through the smokeback produced no effect. This observation in no way conflicts with our experience, as we have found that if there is no mixing of the air and coal, the addition of even 15 or 20 per cent air below the fuel bed is of no value for the control of coke-tree formation. Such additions through the standard smokeback would be difficult when using the regular stoker fan.

I would like to have Mr. Sawyer clear up one point that I noted when reading conclusions 2, 3, and 4. I do not think those statements are intended to be contradictory, but they appear so on the surface, and I believe it would be well to explain them a little more clearly.

C. H. Sawyer (author's reply).—I should have said that when the coal feed was doubled the air ratio was maintained the same. Does that answer your objection?

C. C. Wright.—Yes. I thought that was what you had done.

T. C. Cheasley,* Kansas City, Mo.—I too would like to compliment Mr. Sawyer and his associates on the work they did. We have been doing the same sort of thing for several years, mostly on free-burning coals, to secure better fuel-bed distribution, and we have come to about the same conclusions as to the flare of the tuyeres and the blocking of some of the tuyere openings.

There is one question: What effect did you find on clinker formation by the tests you made?

C. H. Sawyer.—Clinker formation improved as the coke formation was made better; that is, the better coke the stoker made, the more clinker was formed in one week's test. These tests were not run primarily to study clinker formation, and data on clinker formation may be misleading, since the tests were run for an entire week without touching. No clinker was removed until the end of the whole test, and then total clinker for the week was taken as an indication of the way in which the stoker was clinkering. In none of the tests in stoker No. 2 was the clinker as good as it was in the run of tests with stoker No. 1. The better design, which made the coke better, at the same time improved clinker formation.

T. C. Cheasley.—We had the same results. The freer the burning we were able to produce, resulting from control of the "plastic range," the better the clinker formation.

C. H. Sawyer.—If you maintain a smaller fuel bed, with good ignition, you keep the fire down closer and get heat right around the outside of the retort where clinkers form best. If there is much blocky coke, then, in order to supply the demand, there is a fire nearly filling the combustion space. The ash will not stay up there to be clinkered; it falls down anyway, and clinkering will be very much poorer. In most cases good coke formation and good clinker formation go hand in hand.

L. A. Shipman,* Knoxville, Tenn.—I wonder if these coke formations in underfeed stokers have been analyzed as to the fundamental conditions that are favorable to coking and the conditions that reduce the strength of coke formation?

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* Professor of Fuel Technology, School of Mineral Industries, The Pennsylvania State College.

* Sinclair Coal Company.

† Southern Coal and Coke Company.

If so, what conditions were changed when the retort shape was altered, which would have affected the coke structure as analyzed by a by-product coke expert?

In checking through with the pictures, it seemed to me there was a definitely greater movement of the fuel out and over the retort with the sharp turn as compared with the flare turn.

This checks fundamentally, because as the resistance to a horizontal force exerted on a semimobile substance such as coal is increased the vertical movement is increased. As the coal is moved faster out of the retort, what effect would it have on: (1) rate of heating in plastic and preplastic range; (2) time of coking; (3) temperature of coking; (4) oxygen concentration; (5) rate of ignition ratio to rate of burning? This should be considered for the on period and then also for the off period when conditions exist that experience indicates are more favorable for coking.

The authors did prove that size consist as it varied bulk density did affect the coking.

The recording of such phenomena on colored motion "speed-up" pictures gives information that can be repeated until everyone can see and agree on the conditions occurring during the experiment.

I wonder whether in making these changes the authors were trying to make changes to vary the fundamental conditions that cause or do not cause coking.

C. H. Sawyer.—We had many things in mind. I think we had in mind the sort of thing Mr. Shipman means, in our attempts to turn the coal sharper and feed it vertically out of the retort. I am climbing on Mr. Rose's bandwagon here for the vertical screw stoker. I think that is an excellent principle. Apparently the nearer you can feed the coal out of the retort vertically the better behavior there will be from the standpoint of coking. I do not think it is so much a function of time there, because the time—the rate of heating—varies widely in the intermittent operation of a stoker; and coke does not appear to be formed when the coal is fed—that is, when the stoker is on.

In the movie that shows coke treeing, the tree was weakened about every two inches; that is, it was weakened every time the stoker operated. In other words, if the stoker operates continuously we seldom have to worry about solid coke formation; air will blow through and open it up, and it will stay open. This is true of different rates of feeding, and it must be true of different rates of the incoming charge. There is further study indicated there, I am sure; I think there is a series of solid coke formations through long off periods, before the bed above has been filled in, and that is due to a very slow heating process.

Something else enters the picture, which we have not yet had time to consider. We did not really intend to go into design work in this problem. We are coal men. We wanted something we could use, directly, in field service. We practically fell into design. It is hard to get away from it. It is so obvious that 90 per cent of the coking difference noted is due to the difference between the two machines. It seems as though there is a critical region an inch or so below, or just below, the bottom row of tuyeres in the stoker retort—if somehow the heat can be kept away from there during the off periods, and too much heat kept from soaking in there, the stoker will always behave pretty well. When the stoker is on, and there is movement, the incoming coal is disturbed, moved rapidly through the burning zone, and the coke does not form solid and hard. It forms during the off periods by heat soaking down; and I think there is some indication in these tests that heavier, solid cast retorts give stokers a break on that. The heat transferred away by the sheer weight of that metal was enough to prevent its soaking into the incoming coal, to make this hard coke, between the oncoming periods.

The Fly-ash Problem with Domestic Stokers and the Use of Settling Chambers and Firebox Baffles

By T. S. Spicer,* Junior Member A.I.M.E., R. G. Bowman,† Member, and C. C. Wright‡

(New York Meeting, February 1943)

In recent years considerable publicity has been given to the problem of atmospheric pollution by fuel-burning equipment. Legislation has been stimulated and smoke ordinances have been enacted, the provisions of which show a definite trend toward stricter control of the emissions not only from the stacks of industrial fuel users but also from the chimneys of domestic consumers. As a result of war conditions, some slackening in the enforcement of these ordinances may be noted, but this is only a temporary expediency caused by the changes necessary in the distribution of the nation's fuel supplies and difficulties in securing the necessary equipment. Previous trends may be expected to continue in the postwar period.

Largely because of the consumer demand for automatic heat, but partly because of the legislation against smoke emission, there has been a marked increase in the use of domestic underfeed stokers with the past decade. When correctly adjusted and operated, these units reduce the smoke emission materially, so that even the high-volatile bituminous coals may be burned with little or no emission of objectionable smoke. Unfortunately, however, the use of the domestic underfeed stoker introduces another problem; namely, fly-ash.

PRODUCTION OF FLY-ASH

The term "fly-ash" is a misnomer, since the fly material is generally a mixture of small carbonaceous particles varying from green coal to coke, and inorganic particles ranging from fluffy ash to small particles of fused clinker. For want of a more comprehensive name, however, the term fly-ash is used throughout this paper to include all the fly material.

Even with hand-fired furnaces a certain amount of fly-ash is produced, which necessitates periodic cleaning of the boiler flues or furnace radiator, the smoke pipe, and occasionally the chimney itself. With stoker operations the amount of fly-ash is increased many fold, because of the higher velocity of air travel through the fuel bed. Frequently the amount of this fly-ash attains such proportions as to constitute a definite nuisance to the householder and his neighbors and distracts very materially from the otherwise automatic or semi-automatic performance of the stoker.

Virtually all the boilers and hot-air furnaces marketed in the domestic field

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are designed primarily for hand firing and little or no provision is made for removal of fly-ash. Many furnaces are especially deficient in this respect and it is necessary to partially dismantle some of them to remove the fly-ash.

Specific data on the quantities of fly-ash that may be produced by the combustion of coal in underfeed stokers are surprisingly meager. For industrial units operating under normal loads, the figure of about 1 per cent of the total weight of coal burned has been cited by Barkley[1] and others.[2] The exact figure is a function of the furnace design and baffling, of the rate of combustion, and to some extent of the coal burned. Qualitative observations have been made on a variety of clinkering-type domestic stoker installations and the conclusion reached that the fly-ash may amount to as much as a bushel a week. When using the clinkering-type stoker with coals having an ash that is difficult to clinker, there is a tendency even during normal operation for the dry ash to accumulate on the hearth and for a portion of this dry ash to be picked up by the flue gases. Even with coals normally considered satisfactory for this class of stoker, the accumulation of dry ash on the hearth is very pronounced during mild weather because of the low fuel-bed temperatures, and frequently there is excessive production of fly-ash during these periods. From field observations, it is judged that the fly-ash production by the clinkering type of stoker is comparable with that of the ash-removal type although the factors causing the condition may be different in the two cases.

Previously published[3] quantitative data on the fly-ash from an ash-removal stoker operating with bituminous coal in a State College home (Table 1) show that when burning slack coals the amount of fly-ash may vary from about 0.6 to 1.2 per cent of the coal burned. Using sized coals this quantity may be reduced materially, especially if sizes are available that do not degrade seriously in the stoker feed worm. The present practice of using a top size of from 1 to 1¼ in. for sized stoker coal does not decrease the amount of fly-ash appreciably, however, because of the degradation in size before the coal reaches the combustion zone.

TABLE 1.—*Data on Operations of Ash-removal Stoker Using Bituminous Coal*[3]

Coal Source, Seam	Coal Size, In.	Ash in Coal (as recovered), Per Cent	Fly-ash	
			Lb. per 100 Lb. Coal Burned	Per Cent Ash
Fulton-Barnett...	1–0	6.6	0.9	27.4
Fulton-Barnett...	⅜–0	7.7	1.2	25.2
Pittsburgh.......	⅜–10m.	7.8	0.3	58.0
Upper Kittanning	¾–0	8.0	0.6	35.5
Lower Kittanning	¾–⅜	6.6	0.5	49.7
Lower Kittanning	¾–0	7.3	1.0	37.8

Probably more significant than the weight of fly-ash produced in these tests was the volume. For the average home, which consumes approximately 10 tons of coal for the heating season, the volume of fly-ash produced for the slack coals would vary from about 8 to 16 cu. ft. Even with the specially sized coal that produced the lowest weight of fly-ash, the volume amounted to slightly over 4 cu. ft.

At the present time the fly-ash produced in domestic furnaces may collect either in the flue system and chimney or be discharged from the chimney, depending largely upon the design of the particular system and the rating at which it is being operated. In the former case, the deposition of the fly-ash is usually in the most inaccessible parts of the flue system and its removal is a burdensome chore for the householder, who must remove it at frequent intervals in order to prevent gas-backs and to secure satisfactory performance of the heating system. In the second case, it constitutes a nuisance to both the householder and his neighbors by settling out after discharge from the chimney. All too frequently both nuisances exist.

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[1] References are at the end of the paper.

One means for controlling the fly-ash emission from domestic furnaces when using anthracite stokers is suitable baffling of the firebox. A survey of the various types of baffles, details of the method of installing and some information concerning their efficiency has been reported by Anthracite Industries, Inc.[4] When correctly installed for the particular furnace, appreciable reduction of fly-ash is reported. These results substantiate field investigations, which have shown that fly-ash troubles from anthracite stokers are much less frequently encountered when firebox baffles have been installed. Although frequently recommended by baffle manufacturers for use with stokers handling bituminous coal, their possible assistance in reducing the fly-ash nuisance is seldom mentioned, and no scientific data have been found in the technical literature concerning this point.

Until a domestic furnace is designed to take cognizance of the fact that at some future date it will probably be used in conjunction with an underfeed stoker that

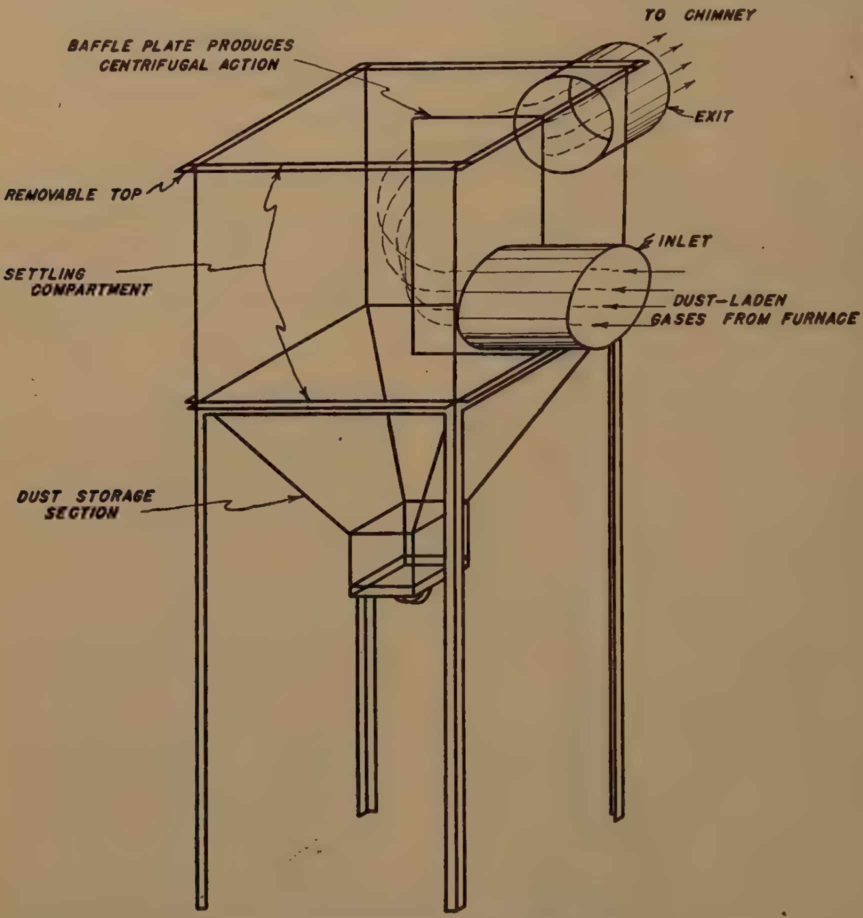

FIG. 1.—PHANTOM VIEW OF SETTLING CHAMBER.

produces fly-ash, it would appear desirable to utilize some type of accessory equipment that will at least in part alleviate some of the householder's difficulties.

METHODS OF CONTROLLING FLY-ASH

In the field of industrial stokers, the fly-ash problem has been fully recognized and a wide range of equipment that solves this problem with varying degrees of success has been made available. The various types of industrial dust-collecting equipment are so well known and have been so thoroughly discussed in the technical literature that little point would be served by an extensive review of the subject. It suffices to point out that the various dust collectors now employed depend in principle upon one or more of the following methods of separation: (1) gravitational, (2) inertial, (3) filtration, (4) sprays and (5) electrical.

To be suitable for domestic use, it is essential that the collector be simple in construction and operation, compact, heat resisting and of such design as not to interfere with the draft by offering excessive resistance to the flow of flue gases. In addition to these requirements, and probably the limiting factor, it must be inexpensive. In view of these limitations, it is apparent that methods 1, 3, 4 and 5 would be impractical owing to space required, cost, or operational difficulties. It is also well known that cyclone dust collectors produce high draft losses, which make their use in domestic installations impossible. It is apparent, therefore, that some form of baffled settling chamber, in which settling, impingement and centrifugal effects are combined, would be the most feasible solution if such a unit could be designed to meet the requirements outlined. Theoretical data for the design of such equipment appear to be nonexistent; so, based upon practical experience with settling-chamber and cyclone collectors, and adhering to the limitations prescribed above, the baffled settling chamber illustrated in phantom view in Fig. 1. was constructed.

Preliminary experiments in the stoker laboratory showed that this settling chamber assisted materially in the control of fly-ash from domestic stokers and indicated the desirability of more detailed information concerning its performance. Tests were conducted, therefore, both in home installations and in the laboratory, to determine the efficiency and practicality of the unit as a fly-ash collector under conditions approaching as closely as possible those obtaining in normal operation of a domestic stoker.

EXPERIMENTAL INVESTIGATIONS

Laboratory Efficiency Tests

The details of construction of the baffled settling chamber can be readily understood from an examination of Fig. 2, which also shows the arrangement of the test equipment used for the laboratory efficiency experiments. In this arrangement the 4-in. pipe serves the dual purpose of producing sufficiently high air velocities to carry all the material tested into the settling chamber and of providing a location for measuring the air flow. Rate of air flow through the settling chamber was controlled by means of a barometric draft control installed between the settling chamber and the fan.

The volume of air delivered to the settling chamber was measured by means of a 4-in. anemometer connected to the end of the pipe leading to the chamber. Prior to the tests, the anemometer was calibrated in place by means of a low-pressure prover. An inclined draft gauge was used to measure the draft loss across the settling chamber.

The materials chosen for efficiency tests included building sand, bituminous coal, bituminous-coal ash, anthracite, anthracite ash and coke. The materials to be tested were obtained from laboratory stock samples, the bituminous-coal ash and coke being obtained from an ash-removal bituminous-coal stoker and the anthracite ash from an anthracite stoker. After reduction, where necessary, to minus 20-mesh in the disk mill, each sample was fractionated, using the Tyler standard screen series, into 10 size ranges, from minus 20 to minus 325-mesh, and the ash content of each fraction was determined by the standard American Society for Testing Materials method.

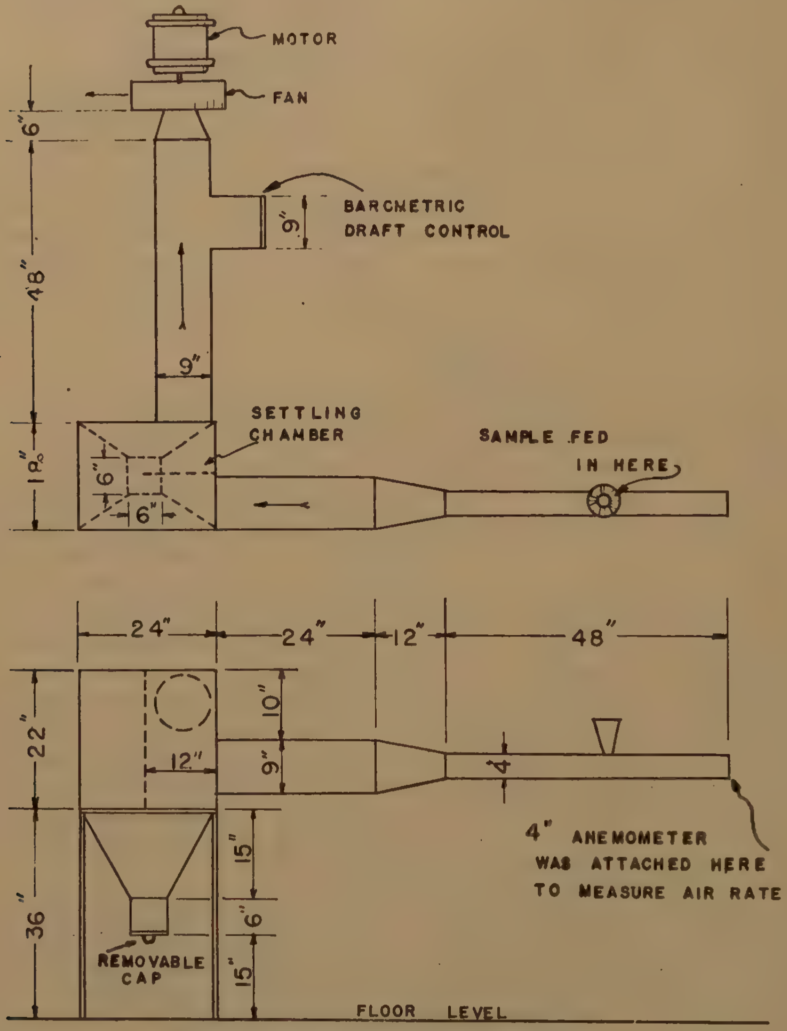

Fig. 2.—Design of settling chamber.

The true specific gravity of the various size fractions of the several materials varies as follows: sand, 2.3377 to 2.6732; anthracite, 1.6692 to 1.6942; coke, 1.7807 to 1.8841. Accurate, true specific-gravity data for bituminous coal, bituminous-coal ash, and anthracite ash could not be secured using the standard A.S.T.M. procedure.

For the efficiency tests two series of experiments were performed. In the first series each size fraction was run separately at the air rate equivalent to that needed for the burning of 20 lb. of coal per hour with 50 per cent excess air at 70°F. (4290 cu. ft. per hour), and at the maximum air rate attainable for the stoker fan used in these tests (7175 cu. ft. per hour). In the second series of tests a composite sample of each of the test materials was made from equal portions of the sized fractions, ranging in size from minus 325-mesh to the size where recovery based upon the previous individual tests was substantially complete. These composite samples were then run at both the low and the high air rates. Samples weighing 500 grams were used for the tests of individual size fractions and 1000 grams for the tests on composite size fractions

TABLE 2.—*Feed and Recovery of Specific Size Fractions of Various Materials*

Size Fraction, Mesh	Weight Feed, Grams	Recovery											
		Sand		Bituminous Coal		Bituminous Coal Ash		Anthracite		Anthracite Ash		Coke	
		Weight	Per Cent	Weight	Per Cent	Weight	Per Cent	Weight	Per Cent	Weight	Per Cent	Weight	Per Cent
At Air Rate of 4290 Cu. Ft. per Hour													
48	500					492	98.4						
65	500					485	97.0						
100	500	497	99.5	495	99.0	465	93.0	493	98.6	493	98.6	495	99.0
150	500	487	97.5	485	97.0	460	92.0	490	98.0	490	98.0	483	96.6
200	500	470	94.0	372	74.4	426	85.2	435	87.0	465	93.0	450	90.0
270	500	314	63.0	265	53.0	355	71.0	330	86.0	445	89.0	430	86.6
325	500	261	52.5	210	42.0	282	56.4	268	53.6	315	63.0	350	70.0
−325	500	182	36.4	150	30.0	205	41.0	214	42.8	217	43.4	217	43.4
Per cent minus 65-mesh recovered.			73.6		65.9		73.0		71.0		81.0		81.0
At Air Rate of 7175 Cu. Ft. per Hour													
35	500					495	99.0						
48	500			498	99.6	482	96.4						
65	500			497	99.4	465	93.0	497	99.4	497	99.4	492	98.4
100	500	495	99.0	493	98.8	455	91.0	494	98.8	492	98.4	480	96.0
150	500	492	98.5	397	79.4	375	75.0	470	94.0	480	96.0	440	88.0
200	500	493	98.6	157	31.4	237	47.4	365	73.0	407	81.4	333	66.6
270	500	279	55.9	135	27.0	209	41.8	187	37.4	342	68.4	286	51.0
325	500	224	44.8	111	22.2	159	31.8	122	24.4	178	35.6	162	32.4
−325	500	143	28.6	72	14.6	122	29.4	75	15.0	133	26.6	126	25.2
Per cent minus 65-mesh recovered.			71.0		45.5		51.9		57.1		67.7		60.0

After the air rates had been adjusted to the desired constant value, the sample was fed by hand into the 4-in. pipe section at the rate of approximately 50 grams per minute. Care was taken at all times to see that the sample was fed smoothly and evenly, with no introduction of large slugs of material. After the material in the chamber had been allowed to settle for a few minutes, the top of the chamber was removed and the entire surface inside the chamber cleaned with a brush, to remove any material clinging to the walls. An air hose and brush were used to remove material that had settled in the feed pipe. The material that had settled in the bottom of the chamber was combined with that from the brushing process and the total was weighed. When composite samples were run through the chamber, the removed material was refractionated and the weight of each fraction was determined.

In Table 2 will be found the data for weight of feed and of recovery of the various size fractions of the several materials tested at the two air rates employed. Table 3 shows similar data for each of the composite samples tested at the two air rates. Table 4 shows the pressure drop across the settling chamber at a series of air rates varying from 2480 to 7070 cu. ft. per hour.

TABLE 3.—*Feed and Recovery of Composite Size Mixtures of Various Materials*

Size Fraction of 1000 Grams, Composite Sample, Mesh	Sand, Grams		Bituminous Coal, Grams		Bituminous-coal Ash, Grams		Anthracite, Grams		Anthracite Ash, Grams		Coke, Grams	
	Fed	Recovered	Fed	Recovered	Fed	Recovered	Fed	Recovered	Fed	Recovered	Fed	Recovered
AT AIR RATE OF 4290 CU. FT. PER HOUR												
48					125.0	107.0						
65					125.0	131.5						
100	166.7	162.2	166.7	167.4	125.0	129.0					166.7	166.3
150	166.7	159.9	166.7	167.4	125.0	119.7	200	198.0	200	192.7	166.7	164.5
200	166.7	162.2	166.7	143.2	125.0	101.1	200	153.9	200	195.0	166.7	151.8
270	166.7	143.5	166.7	117.7	125.0	70.0	200	129.7	200	164.1	166.7	151.8
325	166.7	70.6	166.7	65.8	125.0	64.9	200	107.0	200	135.5	166.7	120.5
−325	166.7	77.6	166.7	47.5	125.0	119.7	200	101.5	200	86.6	166.7	93.3
Total weight recovered...		776.0		709.0		842.9		690.1		773.9		848.2
Per cent minus 65-mesh recovered..............		77.6		70.9		80.5		73.4[a]		81.0[a]		84.8
AT AIR RATE OF 7175 CU. FT. PER HOUR												
35					111.1	97.6						
48			125.0	118.6	111.1	97.6						
65			125.0	118.0	111.1	116.5	143	140.1	143	139.8	143	141.4
100	166.7	162.5	125.0	123.1	111.1	104.5	143	138.2	143	134.9	143	140.0
150	166.7	160.4	125.0	114.9	111.1	103.0	143	128.0	143	127.9	143	128.4
200	166.7	141.2	125.0	40.8	111.1	73.1	143	103.8	143	113.2	143	98.8
270	166.7	95.2	125.0	42.7	111.1	40.6	143	61.2	143	67.8	143	73.4
325	166.7	92.2	125.0	40.4	111.1	26.8	143	43.3	143	39.8	143	45.3
−325	166.7	80.5	125.0	39.5	111.1	86.5	143	22.3	143	75.4	143	59.0
Total weight recovered...		722.0		638.0		746.2		636.9		698.8		686.3
Per cent minus 65-mesh recovered..............		72.2		53.5		65.0		57.9		65.0		63.5

[a] Calculated on assumption that 99 per cent recovery would have been obtained on 100-mesh fractions, which were not included in these composite samples.

TABLE 4.—*Draft Loss across Baffled Settling Chamber at Various Air Rates*

AIR RATE, CU. FT. PER HR.	DRAFT LOSS, INCHES WATER
2,480	0.001
3,430	0.005
4,250	0.007
5,340	0.010
7,070	0.015

Home Tests

During the 1940–1941 heating season, one of the aforementioned settling chambers was installed in home B, a complete description of which has been published previously.[3] The essential features of this installation are: The dwelling is a six-room, brick-veneer, uninsulated house, heated with a gravity hot water system by means of a Weil-McLain model No. A-6-W boiler with 2.64 sq. ft. grate area and fired with a standard 1940 model 20-lb. per hour ash-removal stoker. The location of the settling chamber may be seen in Fig. 3, which shows a photograph of the stoker, furnace, and arrangement of settling chamber.

During the 1941–1942 heating season a second settling chamber of identical design was installed in home A, a description of which has been published previously.[3] The essential features of this installation are: the dwelling is a nine-room, stone-veneer, rock-wool-insulated house, heated with steam by means of an American Radiator Redflash boiler, No. 1-S-9, with 4.09 sq. ft. of grate area, using a standard 1940 model 20-lb. per hour ash-removal stoker. The location and method of installation of the settling chamber were similar to that for the unit in home B.

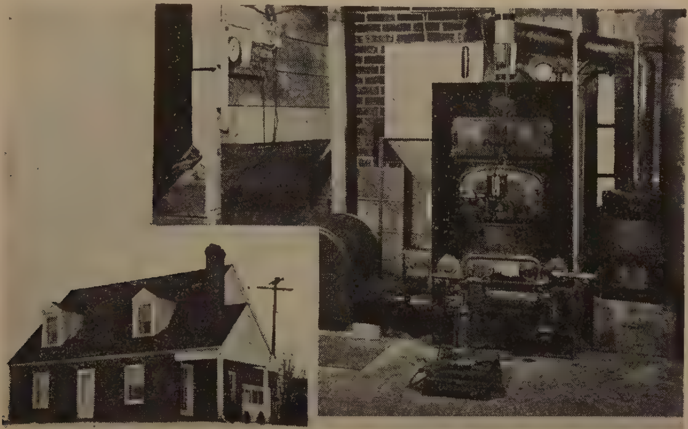

FIG. 3.—SETTLING CHAMBER INSTALLED AT HOME B.

TABLE 5.—*Details of Test Operation in Home Tests*

Data	Home B	Home B	Home A
Test No.	603	604	705
Windbox pressure, in. H_2O	0.9	1.2	0.7
Overfire draft, in. H_2O	0.04	0.04	0.03–0.04
Excess air (start of test), per cent	50	50	50
CO_2 (during "on" period), per cent	7–8	7–8	8–12
Condition of bed	Heavy coke trees	Heavy coke trees	Broken coke mass
Duration of test, days	6	70	18
Total operating time, hr.	47.1	336	82.2
Per cent operating time	29	20	18
Coal burned, lb.	852	6200	1500

At the start of each test the entire furnace, flue system, and settling chamber were cleaned thoroughly. A weighed amount of coal was then burned under the normal operating conditions of the home. At the end of each test the fly-ash was removed from the flue system and the settling chamber, and weighed separately. After determination of the bulk density, the material from both flues and settling chamber was subjected to screen analyses and the ash content of each fraction was determined.

TABLE 6.—*Description of Coals Tested*

Data	Test No. 603	Test No. 604	Test No. 705
Source of coals:			
Seam	Lower Kittanning	Lower Kittanning	Lower Kittanning
County	Clearfield	Clearfield	Centre
Size	¾–0	¾–0	¾–0
Dustproofing	None	Oil	$CaCl_2$
Proximate analysis (as received), per cent:			
Moisture	0.7	0.5	4.2
Volatile matter	21.6	23.1	26.5
Fixed carbon	70.4	66.3	60.6
Ash	7.3	10.1	8.7
Sulphur, per cent	1.36	1.21	1.05
Heating value, B.t.u. per lb.	14,425	13,970	13,154
Ash-softening temperature, deg. F.	2,650	2,800	2,800
Screen analysis, inches or U. S. sieve No.:			
1–¾	0.4	0	0.1
¾–½	13.8	8.4	3.7
½–⅜	9.9	10.6	3.4
⅜–No. 4	20.5	19.6	35.1
Nos. 4–8	13.8	20.0	20.9
Nos. 8–16	12.4	15.2	13.2
Nos. 16–30	11.5	11.2	10.8
Nos. 30–50	7.3	6.8	6.7
Nos. 50–100	4.7	3.7	3.7
Nos. 100–200	3.6	1.1	1.7
Minus No. 200	2.1	3.4	0.7

Details of operation of the stoker are shown in Table 5, screen and proximate analyses of the coals tested in Table 6, and screen and ash analyses of the fly-ash collected from both the flues and the settling chamber in Table 7.

TABLE 7.—*Fly-ash Collected in Flues and Settling Chamber*

Data	Test No. 603		Test No. 604		Test No. 705	
	Flues	Settling Chamber	Flues	Settling Chamber	Flues	Settling Chamber
Total weight for test period, lb.	3.56	5.00	4.19	57.06	4.94	2.35
Total fly-ash per 100 lb. coal burned, lb.	1.00		0.99		0.49	
Bulk density, lb. per cu. ft.	17.6	8.3	17.9	12.4		
Ash content (dry), per cent	54.0	25.0	63.0	33.6	69.5	52.6
Screen analysis, U. S. Sieve No.:						
Nos. 8–16	0.8	1.2	0.3	0.7	0.5	6.0
Nos. 16–30	2.1	9.7	1.1	6.0	1.0	7.7
Nos. 30–50	8.9	22.1	6.0	13.0	4.2	6.0
Nos. 50–100	22.8	20.2	18.9	17.9	12.5	6.6
Nos. 100–200	40.8	19.6	56.8	22.6	32.0	14.3
Minus No. 200	24.6	27.2	16.9	39.8	49.8	59.4
Ash analysis for screen fractions, per cent:						
Nos. 8–16	59.0	11.6	34.0	31.8		
Nos. 16–30	20.3	14.0	20.1	12.6		
Nos. 30–50	37.0	16.7	41.3	17.9		
Nos. 50–100	45.4	24.7	59.0	28.6		
Nos. 100–200	57.2	28.3	66.5	37.4		
Minus No. 200	65.6	34.1	66.6	41.9		

Laboratory Test of Firebox Baffles

For the investigation of the effect of one type of firebox baffle on the control of fly-ash emission from the firebox, a circular flat-type refractory baffle was suspended at a height of 12 in. above the top of the retort of an ash-removal bituminous coal stoker that had been modified to utilize the preoxidation principle of coke-tree control described in a previous publication.[5] The furnace used for these tests was an American Radiator Sunbeam D-524 warm-air furnace with 2.35 sq. ft. of grate area. The stoker-furnace assembly was fitted with a baffled settling chamber identical with those used in previous tests.

Four tests were completed using Lower Kittanning seam coal from the same mine in Cambria County, in ¾ to ⅝-in. stoker and ¾-in. to 0 slack sizes. Using each of the coal sizes, tests were made with and without the firebox baffle installed.

The entire furnace, radiator and settling chamber were cleaned prior to the start of each test; a weighed amount of fuel was placed in the stoker hopper and the fire started. At the completion of each test, all the fly-ash in the radiator and in the settling chamber was collected and weighed. The small amount of fly-ash deposited in the short length of pipe between the radiator and settling chamber was added to and reported as fly-ash in the settling chamber. The experimental data pertinent to these tests are presented in Table 8.

DISCUSSION OF RESULTS

Laboratory Efficiency Tests

The data in Table 2 show that the experimental procedure employed permits a reasonably accurate recovery of material introduced, as evidenced by the fact that better than 98 per cent recovery was secured with the larger size fractions of each of the materials tested. For the individual size fractions of each material, the efficiency of collection decreased progressively with decrease in particle size, as would be predicted from the theoretical equations for both settling chambers and cyclonic separators. At the higher air rates, the efficiency of collection was less than at the low air rates, the effect being most noticeable for the materials of low specific gravity.

The recovery of minus 65-mesh material is better for the composite sample than for the individual size fraction, except with anthracite ash at the high velocity, which suggests that the larger particles tend to knock out of suspension some of the smaller ones. A certain amount of size degradation of all the larger particles occurs either during the screening or during their passage through the system. The fact that the total recovery of minus 65-mesh material is lower for the composite sample of anthracite ash then for the same fractions of sized material can be attributed to excessive size degradation due to high friability of the anthracite ash.

TABLE 8.—*Summary of Fly-ash Results with and without the Use of a Firebox Baffle in a Warm-air Furnace*

Data	Without Firebox Baffles		With Firebox Baffles	
Size of coal, in.	¾–⅜	¾–0	¾–⅜	¾–0
Overfire draft during "off" periods, in. H_2O	0.04	0.04	0.04	0.05
Excess air, per cent[a]	0.4	11.4	0.4	20.6
Per cent CO_2 during "on" periods	12–14	11–13	12–16	10–12
Height of fuel bed above retort, in.	8–10	6–12	4–6	5–9
Condition of bed	Large lumps of coke		Small pieces of coke	
Duration of test, days	5	14	5	14
Cycle of operation	5 minutes "on" 10 minutes "off"			
Per cent operating time	33.3	33.3	33.3	33.3
Weight of fly-ash, lb.:				
Total	6.8	16.8	3.7	14.5
In furnace radiator	4.7	7.9	2.7	8.8
In settling chamber	2.0	8.9	1.0	5.7
Weight of coal burned, lb.	863	2057	857	2162
Fly-ash per 100 lb. coal burned, lb.	0.79	0.82	0.43	0.67

[a] Does not include natural draft leakage through and around ash-removal grate.

At the lower air rate the recovery of all the materials tested was exceptionally good (94 per cent or better) for particles 200 mesh or larger in size, and the over-all collection efficiency, which ranged from approximately 70 to 80 per cent, was reasonably satisfactory. At the higher air rates the collection efficiency for the particles 200-mesh and larger in size was somewhat less satisfactory, but where appreciable size degradation was not encountered the recovery averaged about 90 per cent.

The data for draft loss across the chamber indicate that no difficulty is likely to be experienced in domestic installations operating under normal conditions where satisfactory draft is available. It is recognized that the space velocity of the flue gas will increase with increase in temperature of the gas, but for normal stoker operation in the domestic furnace it appears unlikely that velocities appreciably higher than the range covered in these experiments will be encountered.

Home Tests

The data presented in Table 7 show that the percentage of fly-ash recovered for the two tests at home B amounted to 1 per cent of the weight of the coal fed, while the percentage recovered in test 705 at home A amounted to only 0.5 per cent. No single reason can be given to account fully for this appreciable difference in results, but the following are believed to have been contributing factors:

1. In test 705 reasonably satisfactory fuel beds were obtained during the entire test, while large coke masses were present during all of test 603 and during a large part of test 604. The presence of large coke trees or coke masses tends to produce channeling of the air and causes high localized velocities of flue gases.

2. The percentage of minus 30-mesh material, which is the size range of fly-carbon particles, was less for the coal used in test 705 than in the other two tests. Likewise, the percentage of plus ½-in., which is the size most likely to degrade in the stoker feed screw, was much less in test 705. Thus, the percentage of material of fly-carbon size was probably appreciably lower for test 705 than for either of the other tests.

3. The firebox size was almost twice as large in test 705 as in the tests at home B, and this might be expected to give lower flue-gas velocities. Previous tests using the same coal in both homes, however, gave substantially the same percentage of fly-ash, so there appears to be some doubt as to the influence of this factor.

In comparing the size consist and ash contents of the fly-ash collected in the flues and settling chamber for all three tests, it will be noted that the material from the flues contained an appreciably higher percentage of material in the 100 to 200-mesh size, and that the ash content of the composite from the flues was much higher than that of the composite from the settling chamber. The reason for this is that a large percentage of the fly material blown from the firebox is combustible matter. The part settling in the flues, consisting originally of the larger sized material, is subjected to temperatures high enough to burn much of the combustible, with the result that the ash content is increased. Some of the ash thus produced in the flues is blown out and replaced by fresh fly material, and the process is repeated.

It has been found in a number of installations that where adequate stack draft is available the material in the flues attains substantially an equilibrium condition. In general, this condition of partly filled flues does not appear to impair the flow of flue gases seriously, but occasionally the fly-ash has built up to such a point that it blocked off the flue passages almost completely and gas-backs were produced.

It has been suggested[6,7] that a U-shaped fly-ash trap or a tee should be installed in the smoke pipe from the furnace in order to collect the fly-ash and permit its removal. On the basis of the home tests, it does not appear probable that such a device would be adequate. Such an arrangement would be an improvement over most existing installations, however, where no means is provided for removing fly-ash.

Entirely apart from any characteristics that may be attributable to the fuel, the amount of fly-ash escaping from the firebox is a function of the size and design of the boiler, the kind of baffling in the firebox (if any), the percentage of excess air, and the percentage of "on" time necessary to fulfill the heat demand. Insufficient work has been completed on the effect of the size and design of the boiler to warrant any definite conclusions concerning this problem, although experience indicates that less fly-ash difficulty will be encountered with oversized furnaces. Similarly, experience indicates that less difficulty is encountered when the stoker and boiler are oversized for the heating load.

In discussing the possible use of settling chambers for domestic installations, three specific questions arise: (1) the installation cost of such a device, (2) how much overheating of the basement will be caused by the radiating surface of the settling chamber, and (3) will the average householder be interested in an accessory requiring periodic attention. It is obviously impossible to present quantitative answers to these questions, but the following experiences may be of interest. Each of the units installed in the test homes was handmade at a cost of approximately twelve dollars. On a mass production basis this figure could be materially reduced and because of simplicity of installation the installation charges should be relatively low. Extensive home tests[8] have shown that overheating of the basement will not be produced by the settling chamber even during summer operation of the stoker to provide hot water for household purposes. During the summer of 1942, service hot-water tests were conducted in the basement of home B and the maximum recorded temperature was found to be 78°F. During winter operation no overheating of the basements in either of the test homes had been noticed, although because of the longer firing periods some radiation of heat is apparent. In some respects this extra radiation of heat is advantageous, as it represents recovery of waste heat normally discharged up the stack.

Concerning the home owner's reaction to the use of the settling chamber and his willingness to provide occasional attention, only qualitative information can be offered. The two home owners where test units are installed appear to be well pleased with the performance of the settling chambers and have serviced the units when necessary. Entirely upon their recommendations, several similar installations have been made at State College and at least one stoker service engineer has recommended and installed a number of similar units in other parts of central Pennsylvania.

It must be emphasized that the settling chamber is not the ideal solution to the fly-ash problem, because it is a control method, but until a real solution is found, in the form of a properly designed furnace that eliminates the fly-ash problem, it is believed that the advantages to be gained from the use of this or a similar device will far outweigh any disadvantages that may exist. It is believed, moreover, that the settling chamber will do much to alleviate the fly-ash nuisance experienced by many users of domestic stokers.

Firebox Baffles

The test results presented in Table 8 show a fly-ash emission equivalent to 0.8 per cent of the coal fed when the baffle was not used. This figure is somewhat higher than a home test result of 0.5 per cent obtained a year ago when using $\frac{3}{16}$ to $\frac{3}{8}$-in. coal from the same source on a duplicate stoker. In the former test, however, a hot-water boiler rather than a warm-air furnace was used, the size of the firebox was larger, the percentage of "on" time during the test was appreciably less, and the cycle of operation was quite different, because in the home test the stoker operated on thermostatic demand whereas in the laboratory test the stoker operated on a fixed cycle. Which of the several variables is responsible for the increased fly-ash cannot be established from the data available.

For the test with the baffle installed, the fly-ash emission from the firebox was reduced almost 50 per cent for the stoker size and almost 20 per cent for the slack size coal. Increase in the percentage of excess air for the test with baffle installed partly accounts for the apparently poorer results when the slack size was being burned. Only one type of baffle was tried for this test, but according to data reported by Anthracite Industries, Inc.,[4] this particular type is one of the more effective of the several types tested for the reduction of fly-ash emission.

It should be pointed out that the application of firebox baffles is feasible only where a relatively free-burning fuel bed is possible, as with anthracite; with a few of the freer burning bituminous coals, or with some method of coke-tree control such as was employed in the experiments described. Where heavy coke formations or coke trees are produced by the fuel, laboratory results have indicated that the baffle cannot be used successfully. Similarly, the baffle is not very satisfactory for use with the clinkering type of stoker, because in order to be effective the baffle must be placed in the furnace in a position that will interfere with the convenient removal of clinker. Published data[4] also indicate that the use of baffles is impractical in boilers when the draft resistance across the boiler exceeds 0.10 in. of water. Such a high draft loss across the boiler is rather exceptional if the flues are clean, but the design of many boilers and warm-air furnace radiators makes it possible for the draft loss to exceed this value when fly-ash obstructs part of the flue passes.

CONCLUSIONS

The emission of fly-ash from the firebox of furnaces fired with domestic underfeed stokers varies with a number of factors, which depend upon both the coal and the equipment used. Not infrequently the amount of fly-ash amounts to as much as 1 per cent of the weight of the coal fed to the stoker.

Virtually all domestic furnaces now in use or available to the average householder are designed primarily for hand-fired operation and are inadequately designed and equipped to take care of the fly-ash production normally encountered when used in conjunction with domestic underfeed stokers. Some type of accessory equipment, therefore, is desirable, to assist in the control or removal of this fly material.

A baffled settling chamber small enough to fit conveniently into almost any domestic installation has been designed and its use tested in the laboratory and in the home.

Laboratory tests indicate reasonably high fly-ash collection efficiency for all particles down to about 200-mesh in size, and satisfactory over-all collection efficiency for all particles in the size range normally produced in domestic installations. Draft loss caused by installation of the settling chamber is negligible.

Home tests have shown that the use of this settling chamber or some similar device is both practical and advantageous.

The use of one type of firebox baffle for controlling the fly-ash emission from the firebox has shown that a very material reduction of fly-ash is possible with this device, although the reduction does not appear to be sufficient to overcome all fly-ash difficulties. The limitations of the use of baffles are discussed.

The use of auxiliary devices such as settling chambers or baffles is admittedly not an ideal solution to a problem that involves primarily redesign of the domestic furnace to take cognizance of the fly-ash problem, but such devices will assist the householder in partly overcoming the inherent weakness in existing designs.

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DISCUSSION

(J. E. Tobey presiding)

J. F. BARKLEY,* Washington, D. C.—This paper supplies new and interesting information, particularly to city smoke-regulation engineers. In the days when hand firing predominated, smoke was the principal nuisance that brought on smoke ordinances. The continued replacement of hand firing by the forced-draft stoker has greatly lessened the smoke nuisance but has created a new nuisance—fly-ash. Certain smoke-regulation engineers believe that there should be some type of a fly-ash separating device on all stoker installations. In general it has not been possible to bring much pressure to bear on domestic installations, but many small stokers in apartment houses and elsewhere give considerable nuisance from fly-ash. Installation of expensive and highly efficient dust-separating equipment is not practical for such stokers, but the type of fly-ash separator proposed by the authors could serve exceedingly well.

The authors have supplied test information as to the efficiency of their simple and practicable device. This is of much value to the smoke-regulation engineer responsible for eliminating to the extent possible and practicable the emission of fly-ash from chimneys. Almost no information has been published on the actual performance of the simpler dust-separating equipment.

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* Chief, Division of Solid Fuels Utilization for War, Bureau of Mines. Discussion published by permission of the Director, Bureau of Mines, U. S. Department of the Interior.

As a matter of design, particularly for the somewhat larger installations, some type of gate for dumping the ash from the hopper would be an improvement over the removable cap. It would also be desirable to have one side plate of the device readily removable for easy access to the interior.

L. A. Shipman,* Knoxville, Tenn.—As is usual with such pioneering work, the ideas that the authors present to us for the first time concentrate our thoughts on this subject, which otherwise might be forgotten.

When speaking or attempting to form conclusions from practical tests in furnaces, one finds he has developed a very controversial subject. Because of the thousands of physical and chemical conditions in domestic heating plants, with the hundreds of different coals used and the resultant fly-ash, the number of opinions developed, and the difference of these opinions—all of which may be correct—definitely point out the crying need for more fundamental information to assist in analyzing the conditions.

The test of firebox baffles on control of fly-ash indicates that this baffle was responsible for changing the burning conditions, which may have contributed to the prevention of fly-ash by reducing velocities of gas flow through the fire by increasing the active burning-area as indicated by "small pieces of coke" or by heat radiation, which fused more of the ash to a clinker, or both.

The selection of a method wherein settling impingement and centrifugal force are the contributing factors in fly-ash removal stimulates our minds in thinking of other tests other than particle size, specific gravity, bulk density, on which we have standard or tentative standard methods of determination that may affect the results of the tests and further identify the coal, coal ash, coke, anthracite or anthracite ash.

Tests of grindability, hardness or angle of repose might be an indication of the relative differences of the slowing down of velocities by impingement, as apparently this can affect the results as well as the material the flying particles are impinging against.

I congratulate the authors and hope that this beginning will lead to the design of furnaces and equipment that will not require settling chambers or baffles as *auxiliary* devices.

A. W. Thorson,* Pittsburgh, Pa.—This paper calls attention to one of the problems that should be solved in the course of developing improved coal-burning appliances. The amount of fly-ash emitted by the chimney is dependent, of course, on the type and size of coal and the adjustment of the machine, as well as the nature of the installation. It seems obvious, however, that a completely automatic coal-fired plant should have some provision for avoiding the potential nuisance of fly-ash.

Ideally, a device for collecting fly-ash should: (1) efficiently remove plus 200-mesh material; (2) operate with minimum draft loss; (3) occupy a minimum of space; (4) permit easy removal and disposal of collected material; and (5) be an integral part of and incorporated within the heating unit. The authors' design appears to accomplish the first two and fourth objects reasonably well. It is hoped they will continue their work on this much neglected subject.

A vast amount of work has been done to develop industrial dust collectors, and many data and much information on design are available. Careful study of this field might point to applications of a highly efficient type to the domestic heater. Such a collector would save space, which is a very important factor, particularly if the fifth objective is to be fulfilled.

On page 242 the authors discuss the abnormal size distribution of material collected in the heater flues. They explain that a higher percentage of 100 to 200-mesh size was collected in the flues than in the collector because the coarse portion settling in the flues is subjected to temperatures high enough to burn much of the combustible. This material is then blown out as it is consumed, leaving a high percentage of finer material. In my opinion, the temperatures in the flues do not approach the 1000° to 1200°F. necessary to again produce ignition and combustion of the particles blown from the fuel bed. The abnormal size distribution may be explained, however, in the following manner: The air through the fuel bed carries with it particles varying from the maximum density the air velocity will carry down to the finest material present. Part of the solids will collect on the flues and in corners and pockets where local velocity is low. After a time these fill up, and the gases scour a definite path, after which time little additional material will collect. The remainder of these solids will deposit in the settling chamber, chimney pipe or elsewhere, or will pass out of the stack. Neither the part in the flues nor that in the settling chamber will be representative. If we dump the two collected quantities together and obtain a sieve analysis, the composite should represent that material leaving the fuel bed, less, of course, the fine sizes going to the stack. Accordingly, the writer recomputed Table 8, as shown in Table 9, to obtain composite size distribution. The two portions and the composite for test 604 are plotted on Fig. 4. This test was chosen because the quantity of fly-ash collected was much larger and should be subject to the least experimental error.

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* Assistant to the President, Carnegie Institute of Technology.

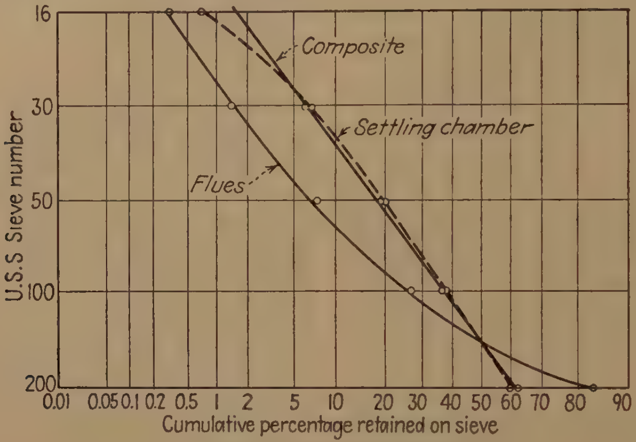

FIG. 4.—SIEVE ANALYSIS, TEST 604.

Although the size distribution of the two portions forms curves, that of the composite quantity forms a straight line on probability

TABLE 9.—*Recomputation of Table* 8

Test No.	603					604					705				
Place	Flues		Settling Chamber		Total	Flues		Settling Chamber		Total	Flues		Settling Chamber		Total
Total weight collected, lb.	3.56		5.00		8.56	4.19		57.06		61.25	4.94		2.35		7.29
Sieve Analysis	Per Cent	Cum. Per Cent	Per Cent	Cum. Per Cent	Wt. Avg. Cum. Per Cent	Per Cent	Cum. Per Cent	Per Cent	Cum. Per Cent	Wt. Avg. Cum. Per Cent	Per Cent	Cum. Per Cent	Per Cent	Cum. Per Cent	Wt. Avg. Cum. Per Cent
−8 + 16	0.8	0.8	1.2	1.2	1.0	0.3	0.3	0.7	0.7	0.7	0.5	0.5	6.0	6.0	2.3
−16 + 30	2.1	2.9	9.7	10.9	7.6	1.1	1.4	6.0	6.7	6.3	1.0	1.5	7.7	13.7	5.4
−30 + 50	8.9	11.8	22.1	33.0	24.2	6.0	7.4	13.0	19.7	18.9	4.2	5.7	6.0	19.7	10.2
−50 + 100	22.8	34.6	20.2	53.2	45.3	18.9	26.3	17.9	37.6	36.8	12.5	18.2	6.6	26.3	20.8
−100 + 200	40.8	75.4	19.6	72.8	73.8	56.8	83.1	22.6	60.2	61.8	32.0	50.2	14.3	40.6	47.1
−200	24.6	100.0	27.2	100.0	100.0	16.9	100.0	39.8	100.0	100.0	49.8	100.0	59.4	100.0	100.0

paper, thus indicating a normal distribution of particle sizes. This indicates that the particles that were expected to be found in the flues, but were not, were actually found in the settling chamber.

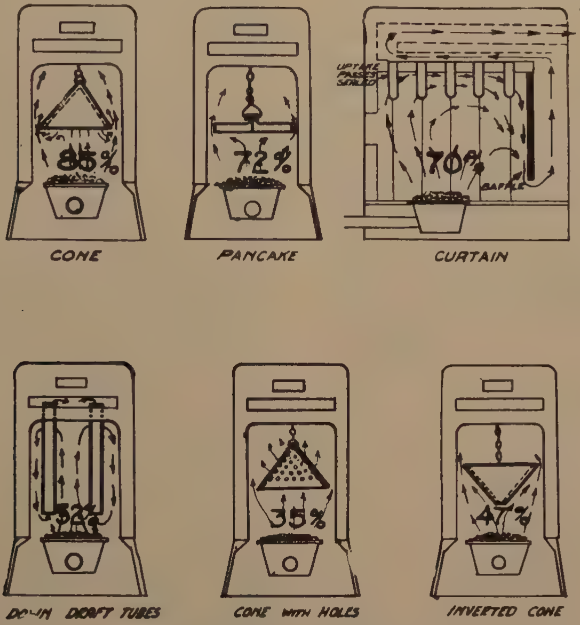

Fig. 5.—Domestic stoker baffles, with relative effectiveness in reducing fly-ash deposited in passes.

A seventh type of baffling-flue pass bricks (not shown) was found to reduce fly-ash deposits in the passes to 41 per cent of normal; i.e., it was 59 per cent effective.

It would be interesting to have similar plots obtained by use of different coal types and sizes. This information would add materially to our knowledge of the material we are called upon to collect, which in turn would enable us to design the ideal dust collector for the domestic heater.

A. J. Johnson,* Philadelphia, Pa.—The authors undoubtedly have made a valuable contribution in connection with an almost entirely neglected phase of stoker performance. Both the measurements and analyses are timely and will be the basis for much future research toward the ultimate solution of this vexing problem.

However, with regard to the use of auxiliary equipment for collecting fly-ash, it is to be hoped that this work can be continued to present further information leading to the lessening of fly-ash at the source—in the furnace.

With anthracite as the fuel, the principal annoyance of fly-ash is as the result of its accumulation in the flues and passageways of the heater; therefore a settling chamber between the heater and the chimney will not alleviate the real complaints.

Suggestions for possible solution, or at least subjects for further research, may be outlined as follows, with emphasis upon the fact that these suggestions are primarily intended to apply to anthracite-burning heaters:

Designing Heaters for a Minimum of Fly-ash

In designing new heaters, the potential formation of fly-ash should be given serious consideration.

The basic principles of sudden changes in gas direction and velocity can be built directly into the furnace to throw a major percentage of the fly-ash directly into the ashpit or back on to the fuel bed. This can be accomplished by appropriate design to secure the effect of either curtain baffles, downdraft tubes, or bell baffles.

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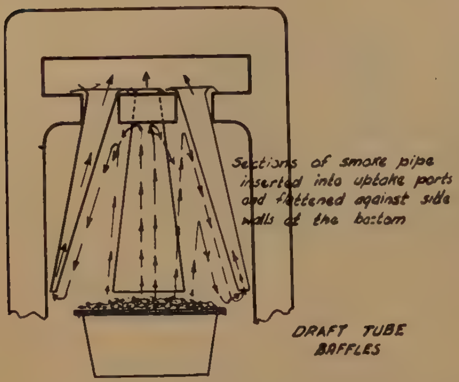
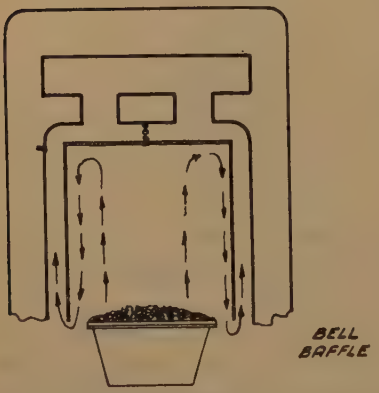
* Mechanical Engineer, Anthracite Industries, Incorporated.


Sections of smoke pipe inserted into uptake ports and flattened against side walls at the bottom

DRAFT TUBE BAFFLES



BELL BAFFLE

FIG. 6.—IMPROVEMENTS IN BAFFLE APPLICATION.

In steel boilers, and to a somewhat lesser extent even cast iron, the use of small tubes or passageways having relatively high gas velocities has been shown to materially reduce the fly-ash that will be deposited.

In brief, a new heater should be designed to trap a maximum amount of fly-ash in the furnace; to deposit as little ash in the flues and tubes as possible; and, finally, it should have an integral collecting bootleg just ahead of the breeching, with an easily operated, clearly marked cleanout door or port. Such a bootleg should be of ample size for a full year's accumulation of fly-ash.

Baffling Existing Heaters to Minimize Fly-ash

Existing equipment, naturally, cannot be changed so radically to minimize fly-ash. However, as the authors have shown, the use of furnace baffles can effect a very material reduction. Their tests to date report experiences with only one type of baffle. This reduced the fly-ash from 0.81 to 0.55 lb. per 100 lb. of coal (average of two tests). The baffle used was of the pancake type, which we had previously found to be the second best of seven types tested.

Fig. 5 shows these seven types of baffles in the order of their efficiencies. The figures on each baffle refer to the percentage of the fly-ash that the baffle held back from accumulating in the heater passes; in other words, the direct measure of the effectiveness of the baffles. That these figures are high as compared with the authors' is probably because we measured only the ash in the flues and not that which passed through the heater. However, the relative position of the several baffles would not be affected.

Subsequent to these tests at least two improvements have been made in baffle application to further increase their effectiveness. These are shown as Fig. 6. The first, a bell baffle, is not only highly effective but also has the advantages of forcing the gases to scrub the heater walls and of transmitting additional radiant heat from the hot baffle to the heater surfaces. Unfortunately, it has the serious disadvantage of having to be custom built; is rather difficult to install; and is hard to design so that the fire can be built and inspected readily. Probably its most promising future application is in new boilers, where these disadvantages can be overcome.

The second improved baffle is of the downdraft-tube type, but differs from that tested in the important respect that the tubes have been carried as close to the fuel-bed level as possible and placed back against the heater side walls. These features prevent the fire from "looking into" the tubes and tend to force all furnace gases into a circuitous course. The construction and installation of these baffles is simple. In fact, many have been fashioned of ordinary smoke pipe, fastened to the uptake openings by being multiple slitted, then bent in place into flange form, and with the lower ends flattened against the heater walls.

In conclusion, it should be stressed that the use of baffles as a solution to the fly-ash problem has the important additional advantage of improved heater efficiencies and outputs.*

J. D. Doherty,† Pittsburgh, Pa.—The authors' investigations of the fly-ash problem with a moving-grate ash-removal stoker for bituminous coal and their frank discussion of the fly-ash problem should be very helpful toward a final solution of the problem.

It may be of interest to compare the amount of fly-ash deposited in the boiler flues with the stoker used by the authors and that found with the conventional type of domestic bituminous-coal stoker; that is, the under-feed single-retort stoker in which the ash is removed as a clinker, and which has no agitation of the fuel bed. At the Stoker Coal Research Laboratory of the Koppers Company, a variety of bituminous stoker coals have been tested with a small clinkering-type stoker installed in a cast-iron sectional boiler, as described by Knox and Doherty.‡ The amount of fly-ash removed from the boiler after having burned about 520 lb. of coal (at normal air settings and a coal feed rate of 15 lb. per hour) ranged from 0.08 to 0.24 lb. per 100 lb. of coal burned. The authors of the paper under discussion report that with a ¾ to ⅜-in. Cambria County coal burned under similar conditions in the ash-removal stoker the fly-ash deposit was 0.79 lb. per 100 lb. of coal burned. These figures indicate that with the clinkering stoker the fly-ash deposit is only a fraction of that produced with the ash-removal stoker. This confirms the generally accepted belief that the agitation or disturbance of the fuel bed, especially when the stoker is running, greatly increases fly-ash.

Investigations in the Koppers laboratory indicate that fly-ash deposits with the clinkering-type stoker can be decreased by decreasing the air rate, decreasing the coal feed rate with corresponding decrease in air rate, decreasing the overfire draft, and avoiding as far as possible agitation of the fuel bed, especially when the stoker is running.

T. S. Spicer, R. G. Bowman and C. E. Wright (authors' reply).—The authors appreciate the thought and effort that the several discussers have demonstrated and the numerous useful suggestions that are made which contribute materially to a more thorough understanding and recognition of the fly-ash problem.

Mr. Barkley's suggestion of a grate for dumping the ash would be especially useful on larger units than those normally required for purely domestic heating plants.

Little can be added to Mr. Shipman's remarks, because unfortunately the data suggested as desirable were not secured. In further work it would be of interest, however, to secure such data and determine the correlation.

Mr. Thorson's suggestion concerning the size distribution of the composite fly-ash in flues and collector is most interesting and may to some extent explain the observed phenomenon. In the home test reported, however, it was observed on numerous occasions that the fly material was, as described in the paper, actually burning in the flues. Based upon numerous measurements of the temperature of flue gases leaving a number of furnaces and boilers, the authors believe it would be a rather unusual domestic installation where the temperature in the flue passes of the boiler does not exceed 1000°F. during the on periods.

Mr. Johnson's discussion and data are most valuable and are a definite contribution to the meager literature on the use of baffles.

The authors are unable to agree with either the comparisons made or conclusion reached by Mr. Doherty. A comparison of the fly-ash collected only in the flue passes of a cast-iron boiler is made against the fly-ash collected in the radiator, smoke pipe and dust collector and a warm-air furnace and the conclusion reached that the ash-removal stoker produces more fly-ash. A comparison between the data of Table 7 for fly-ash collected in the boiler flues only and that of the paper by Knox and Doherty‡ would appear more logical and would lead to the opposite conclusion. Such a comparison would be unjustified, however, because cycle of operation, type of firebox, character of the coal, excess air and several lesser variables influence the results.

Comparative tests have been made by the authors using the same coal on a fixed cycle of

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* References: Baffles for Stoker Fired Boilers. *Bull.* L38, Anthracite Industries, Inc.

† Koppers Coal Division, Eastern Gas and Fuel Associates.

‡ W. Knox and J. D. Doherty: Research on Coal for Domestic Stokers. *Trans.* A.I.M.E. (1942) **149**, 254.

TABLE 10.—*Fly-ash Data for Clinkering and Ash-removal Types of Stokers*

Size Coal, In.	Firebox Baffle	Fly-ash Collected, Lb. per 100 Lb. Coal Burned			
		Clinker-type Stoker		Ash-removal Stoker	
		Flue Pass	Total	Radiator	Total
$\frac{3}{4} \times \frac{3}{8}$	No	0.17	0.25	0.19	0.33
$\frac{3}{4} \times \frac{3}{8}$	Yes	0.17	0.27	0.39	0.79
$\frac{3}{4} \times 0$	No	0.29	0.93	0.38	0.82
$\frac{3}{4} \times 0$	Yes	0.26	0.61	0.40	0.67

operation—5 min. on, 10 min. off—in a clinkering type and in an ash-removal type of stoker, both of which were modified to incorporate the preoxidation principle. The clinkering-type stoker was installed under a standard boiler and the ash-removal type under a standard warm-air furnace. Approximately one ton of coal was burned in each of the tests, the data for which are shown in Table 10.

These data indicate that the amount of fly-ash from the two types of stokers is probably of the same magnitude when the clinkering-type stoker is operated on a sufficiently heavy cycle to ensure clinkering of the ash. When on a mild cycle, such as frequently is encountered during fall and spring operation, experience indicates that much more fly-ash may be expected from the clinkering-type stoker unless the ash of the coal employed is unusually easy to clinker.

Selection of Coal for Carbonization

By B. P. Mulcahy*

When the phrase "selection of coal for carbonization" is used, there is always the implied continuance of thought "to make good coke." The reason for this, of course, lies in the fact that, in almost all instances, when coal is carbonized coke is the principal product. Under certain conditions the yield of gas may also be a factor, but even here it must be secondary to the coke yield, for in every instance coke represents the largest single product of the process.

The importance of coke to the by-product industry has been considerably enhanced during the past 20 years because of the continuously decreasing financial returns of all the by-products. This fact will account, in part, for the rebirth of interest and investigation in the carbonization process during the past 10 years.

Many papers have been written on this subject, with naturally divergent thoughts because of the diversity of interests, but in each instance the common aim was the same—the making of better coke.

Scope of Investigation

In perusing this literature, the writer was struck with the constantly recurring imposed condition of asking for better coke, but with the distinct unwillingness to make any concession to temperature of carbonization.

When there is a high demand for coke, the only means by which the industry can increase the quantity is to shorten the coking time by an increase in temperature, or to increase the quantity of low-volatile coal, or both; and since in many instances the percentage of low-volatile coal can be increased only in small quantities, because of factors to be discussed later, the big increase must come from shortening the coking time. When this is done most of the improvement accomplished by the selection of the coals in the mixture is lost.

It is not felt necessary in this paper to discuss the advantages of minimum ash and sulphur and other materials in coal. The value of minimum contents of these materials in coke is well established. When the use of coals high in these contents is dictated, the selection of coals, as we will consider it here, cannot apply to full advantage.

The scope of this paper, then, will be to discuss the factors of inherent coal characteristics and operating procedures that influence coke quality, for they are fundamentally related and both must be considered if we are to attain our objective of better coke.

A study of either must embrace a study of both simultaneously. To properly evaluate the influence of the coal and to help in the selection, we must first be able to appraise the coke, and interpret fairly the influence of the process in all of its aspects.

Appraisal of Coke

To refer simply to the quality of coke gives no indication of what is meant, for the term is too general in its implication. An analysis of a coke is divided usually

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into two groups of characteristics, chemical and physical.

We are all familiar with the terms in the usual chemical or proximate analysis, volatile matter, fixed carbon, ash and sulphur, and what they indicate, and these in general we will not discuss. It is the physical analysis of the coke that we are concerned with, for these factors in most instances exert by far the greatest effect upon the performance of the coke. The principal factors in this category are as follows: size, strength, weight per cubic foot, porosity and cell size, and of these size and strength alone will give an index of coke quality more important in general than all of the other factors combined, for they represent to a very large extent the composite effect of the coal and the process.

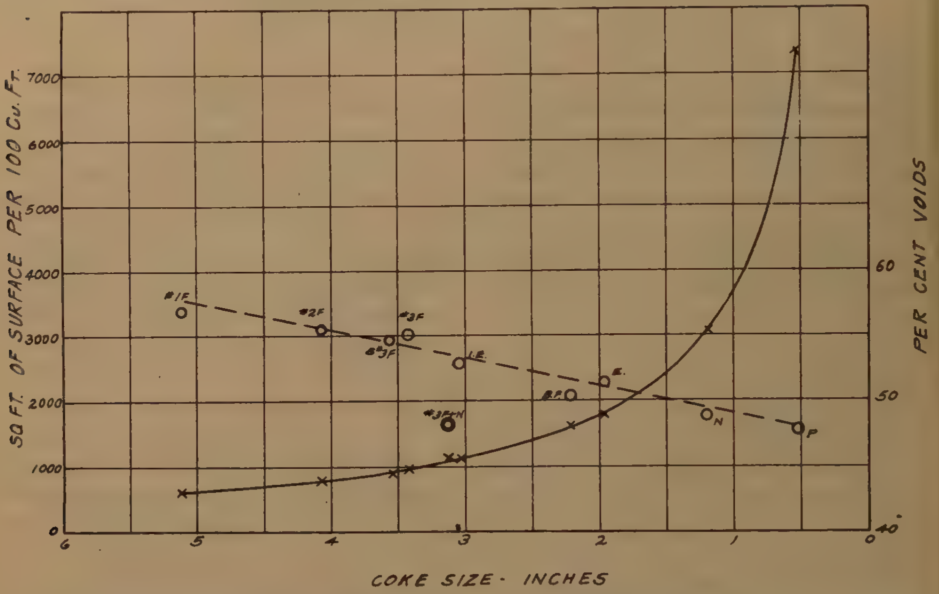

FIG. 1.—COKE SIZE VS. AREA AND VOIDS.

The broken, straight line indicates coke size vs. voids; the solid asymtotic curve indicates coke size vs. surface.

TABLE 1.—*Coke Size vs. Voids and Surface Area*

Size	Solid Wt. per Cu. Ft., Lb.	Actual Wt. per Cu. Ft., Lb.	Coke, Per Cent	Voids, Per Cent	Average Size, In.	Surface Exposed (Assuming Cubes)	
						Sq. Ft. per 100 Cu. Ft.	Sq. Ft. per 100 Lb.
No. 1 foundry	56.4	24.30	43.1	56.9	5.12	604	24.9
No. 2 foundry	56.4	25.12	44.5	55.5	4.08	785	31.2
Special No. 3 foundry[a]	55.3	25.00	45.2	54.8	3.57	909	36.4
No. 3 foundry	56.4	25.35	44.9	55.1	3.42	946	37.2
No. 3 foundry[a] and nut	55.3	28.60	51.7	48.3	3.24	1,150	40.2
Ind. egg	56.4	26.60	47.2	52.8	3.051	1,110	41.7
Blast furnace	56.4	28.00	49.6	50.4	2.221	1,604	57.3
Egg	56.4	27.40	48.6	51.4	1.973	1,777	64.6
Nut	56.4	28.80	51.1	48.9	1.199	3,070	106.6
Pea	54.5	28.40	52.1	47.9	0.51	7,350	259.0

[a] Tests on coke of wide spread in size consist.

In the following discussion it is neither inferred nor implied that the other factors are unimportant. The object is to clear the stage of all nonessentials so that the components of the problem may be evaluated properly. The total ash and sulphur contents of coals are controlled outside the coke plant, and therefore are intended to have no influence in this consideration. The size of the ash particle, however, is well within the control of the plant operator, and he who willfully disregards its influence by not crushing his coal sufficiently fine cannot be seriously interested in good coke structure.

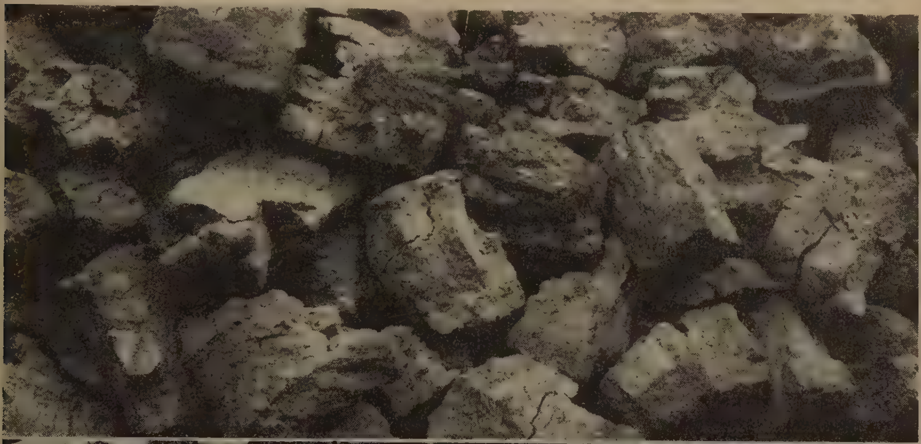
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Fig. 2.—Large (No. 1) foundry coke.

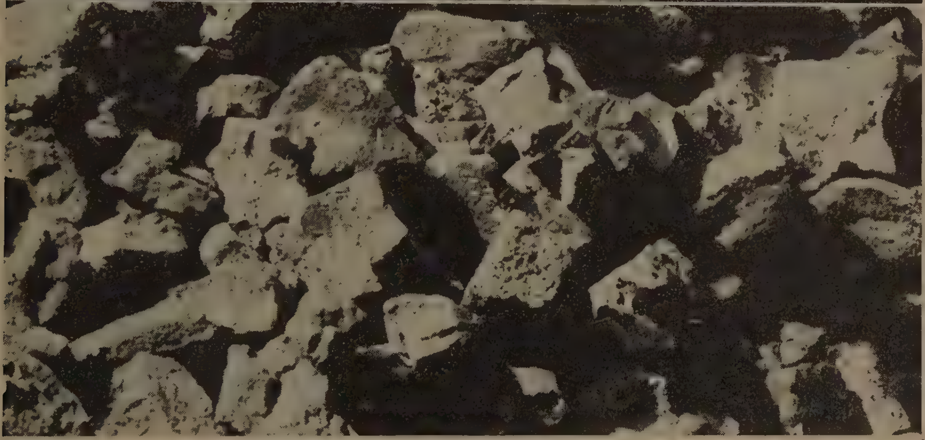
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Fig. 3.—Medium (No. 2) foundry coke.

Size of Coke

Size and strength of coke are closely related, for the size of a coke from any coal mix, carbonized at a given rate, is a direct function of the fissure development in the coke and the resistance of this same coke to degradation under physical handling is a function of the same property. This point will be illustrated later.

First, consider size alone. Fig. 1 shows the percentage of natural voids between the pieces in various sizes of coke. The data for the curves are contained in Table 1.

The index used is a weighted average size obtained by complete screen analysis. As the curve in Fig. 1 indicates, the greater percentage of voids occurs with the largest coke size, which means that a given volume will contain a smaller quantity, by weight, of a larger sized coke. The second curve on this chart, coke size vs. surface area, is a composite of the voids and the area per pound computed to the same volume. In computing these areas, cube surfaces were used. Spheres could have been used to show the relative area change with size. This curve gives striking evidence of the fact that surface area increases in direct proportion to the size decrease; in other words, a 2-in. coke has twice the surface area of a 4-in. coke.

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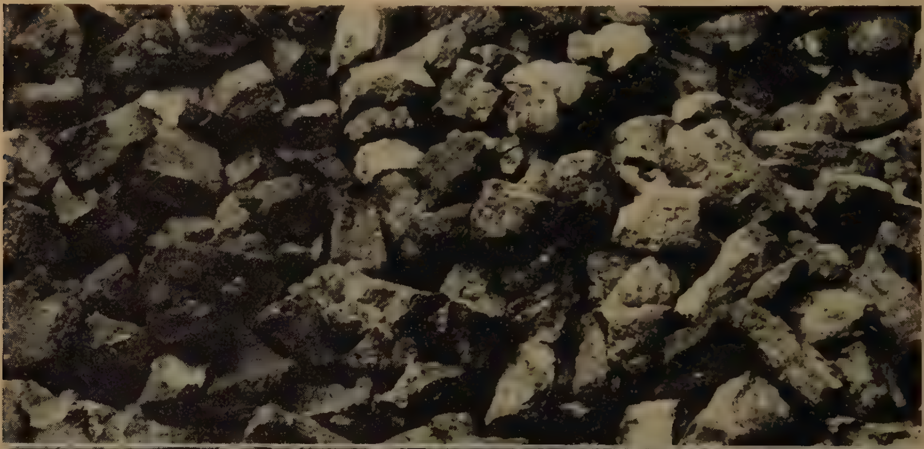

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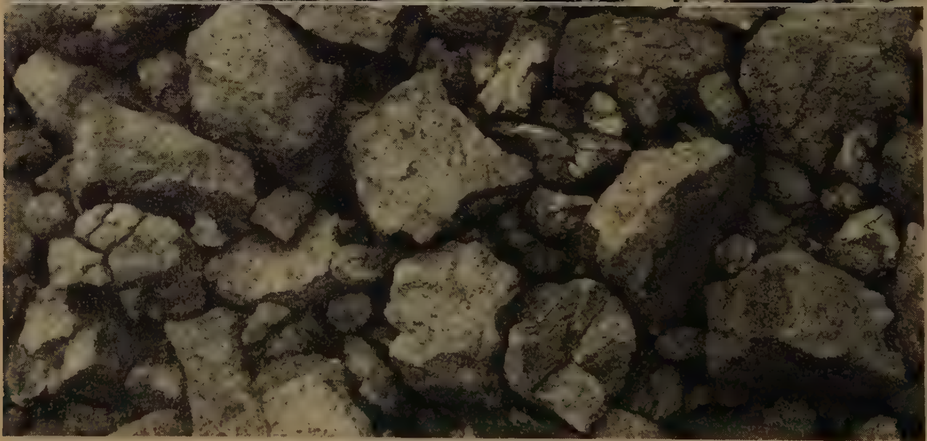

Fig. 4.—Small (No. 3) foundry coke.
Fig. 5.—No. 3 foundry and nut coke.

The full effect of this factor cannot be appreciated, however, without going a step farther and considering what happens when a coke has a wide spread in size consist. A No. 3 foundry coke (4 to 3 in.) was loaded in an 8-cu. ft. box and its weight per cubic foot was determined. Nut-sized coke (1½ to ¾ in.) was added to this coke to its saturation point. The weight per cubic foot of the mixture was again determined. These tests are listed in Table 1.

In these tests, the void space decreased 6.5 per cent (from 54.8 to 48.3 per cent), the weight per cubic foot increased 3.8 lb., or 14.5 per cent, the average particle size decreased 0.19 in. and the surface area correspondingly increased 23 per cent (Table 1). While this was a prepared sample, the same effect takes place with a weakly structured coke, as it breaks in handling, and often to a greater extent than the example used.

Figs. 2, 3, 4 and 5 show the average size consist and appearances of foundry sizes No. 1, No. 2 and No. 3 and the mixture of No. 3 with nut. They give a general idea of the sizes of voids encountered in coke ranging from 3 to 6 in. when the pieces in each size range are fairly uniform.

The value of this relationship lies in the interpretation of its effect. It can be stated that, other conditions remaining the same, the rate of combustion of any coke is a function of the type of carbon present and of the particle size. In other words, a coke of 2-in. average size with a residual volatile matter content of 2.0 per cent will burn faster than a 2-in. coke with only 1.0 per cent volatile matter content. This is true because the influence of the additional 1.0 per cent volatile matter lowers the ignition point of the coke appreciably, and the oxygen surrounding the coke particle is more rapidly consumed in the reaction with the carbon. With this same coke also the rate of decomposition of the CO_2 is faster—which means that after the O_2 is consumed or depleted the CO_2 decomposition can proceed at a faster rate; temperatures, of course, permitting. The effect of a coke of this nature is to narrow the zone of maximum temperature in the bed where O_2 exists, and the secondary reaction of CO_2 decomposition tends to produce lower temperatures over a broader zone above the combustion area.

This same effect of change of rate or reaction is also a function of coke size, because of the differences in exposed coke area. The writer firmly believes that this effect of size is more important than the change in character of the carbon (encountered in normal carbonizing practice) in the influence on over-all temperatures and reaction rates in the combustion of coke,

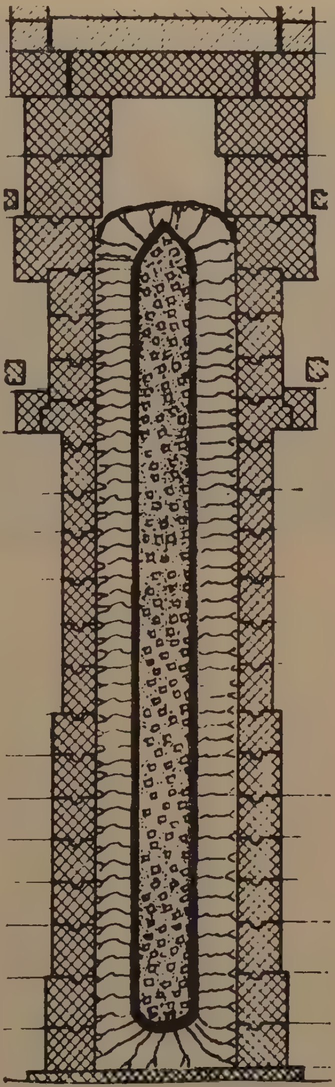

Fig. 6.—Coal charge in oven.
Adapted from Morgan.[3]

and, further, that a coke with a wide range of sizes cannot be controlled in its combustion rate nor give the same thermal efficiency as a similar coke uniformly sized. This reference to combustion rate must not be confused with the inherent combustion characteristics due to type of carbon. The meaning here is the rate of combustion of coke under actual operating conditions. Invariably when the size of coke is decreased, either by design or by accident, the maximum temperature zone is shortened and lowered closer to the point of oxygen entry into the bed.

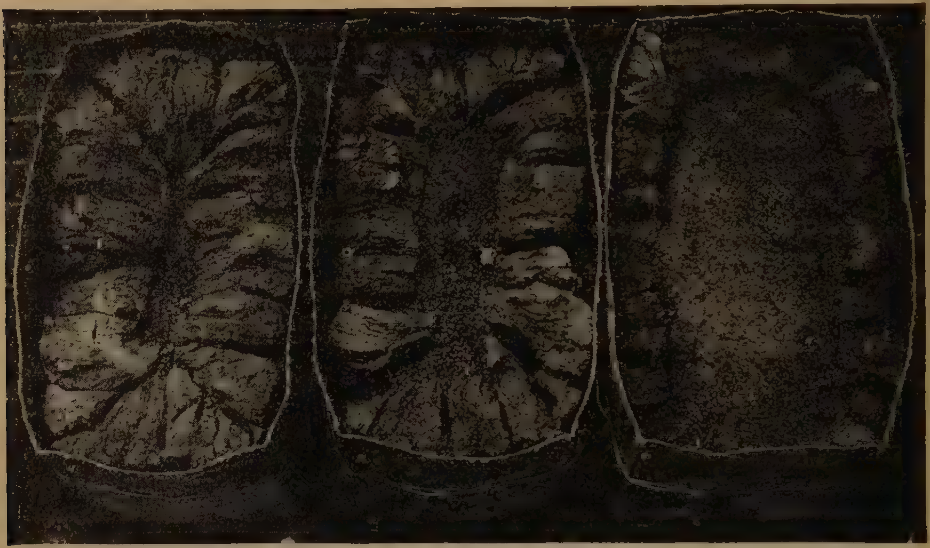

FIG. 7.—BOX TESTS IN EMPTY OVEN.[4]

Brown[1] undoubtedly had this in mind when he suggested an increase in the volatile matter content of coke to 1.0 or 1.5 per cent, which we know will produce a larger coke with fewer fissures.

The control of the volatile matter content of coke is entirely in the hands of the plant operator because it is a direct function of carbonizing temperature. Some temperature variables in different sections of the oven are inherent, because of oven design and operating features, and these will account, in part, for some swings in volatile matter content, but the greater portion of the coke can be held within certain practical low limits.

The wide range of sizes produced and their consequent extreme diversity of industrial use has demonstrated time and again the effect of size upon combustion rate.

Another coincident effect of smaller coke is the decrease in permeability of the bed. In the instance of smaller coke, the bed density increases, and under the same pressure conditions a greater inequality of distribution of gases occurs, causing local increases of flow, particularly around the periphery. This situation is materially aggravated when the coke contains a wide range of particle sizes and the void space therefore is reduced.

The conditions enumerated above are fundamental and influence the thermal or metallurgical conditions, regardless of the process burning the coke. An atmosphere of either high CO or CO_2 can be created with a good sound coke, properly sized, by regulating the quantity of air

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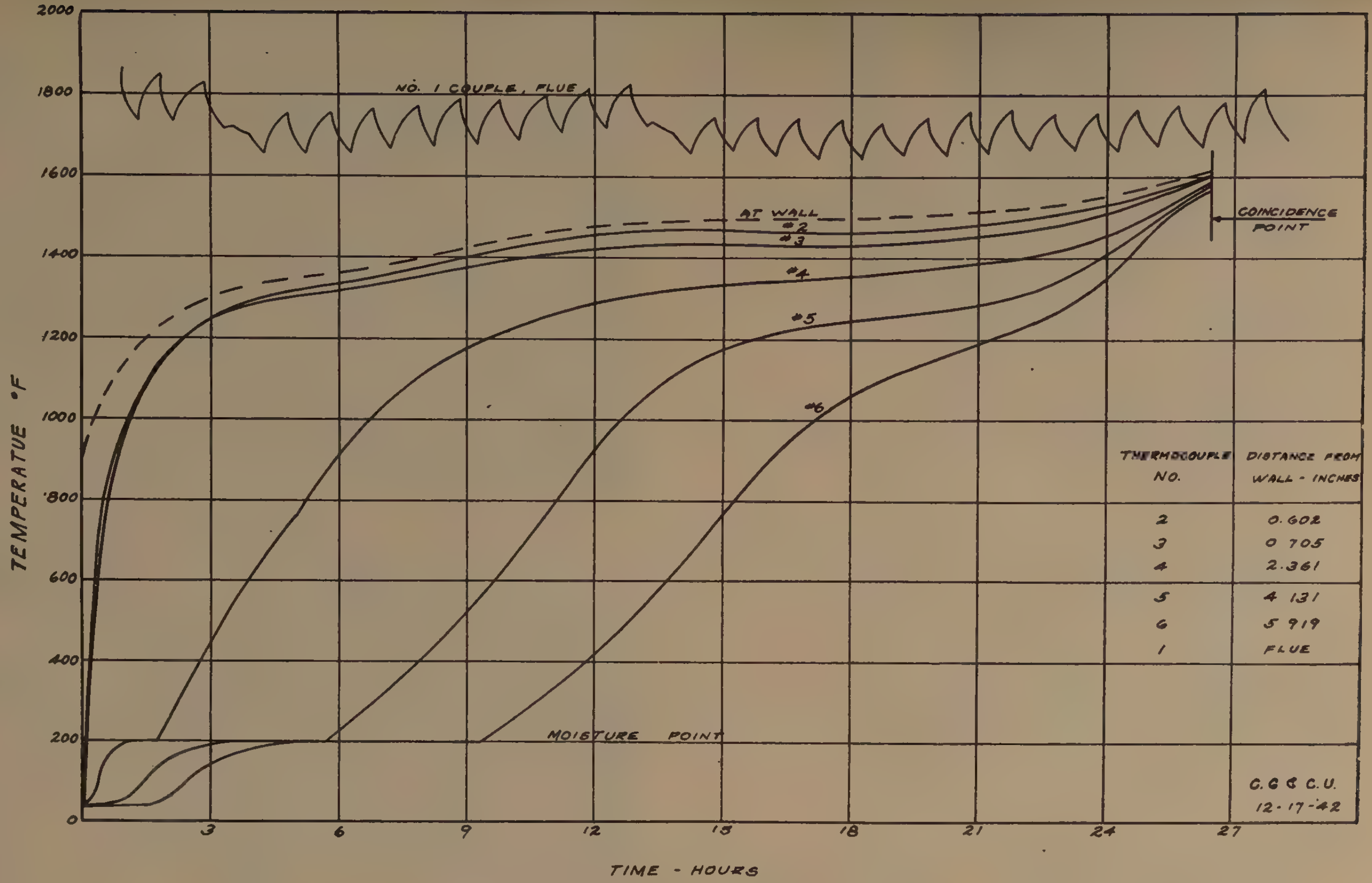


FIG. 8.—TIME-TEMPERATURE CARBONIZATION CURVE.

used. This statement, of course, opposes the belief that a "fast-burning" coke is a requisite for blast-furnace operation, which usually connotes free burning characteristics due to a high residual volatile matter content. The latter is neither necessary nor advisable, for greater thermal efficiency can be accomplished by the use of a firm, regularly sized coke favoring proper gas permeability and with uniform surface exposure.

Operating Factors Affecting Coke Structure

Operating factors affecting coke structure are generally well known but unfortunately many operators disregard their influence or, because of certain conditions beyond their control, are unable to control their individual effects. A brief review of these factors, however, is necessary in this discussion because their influence on coke structure can produce the same objectionable results as different coal characteristics.

Control in Mixing Two or More Coals

Control in mixing two or more coals is usually accomplished by means of volume mixing; that is, gate settings above a belt regulate a predetermined quantity of each coal. In recent years this procedure has been modified by the use of weight meters, with which coals are mixed by weight, and which appreciably improves the control of this factor. In our own experience with volume mixing, formerly we had a variation of ±8 per cent from the desired quantities, while now, with weight-mixing the variations are approximately ±2 per cent.

Fineness of Grinding

In grinding, there are several possible effects. First, the degree of fineness influences the coke structure because of the melting or fusing properties of the individual coals and the different materials in the same coal. Fine grinding favors a more intimate mixture, but at the same time may result in low bulk density and thereby decrease oven capacity. When oven capacity is the main objective, and coarse grinding is resorted to, with consequent improper blending, the quality of the coke must suffer. On the other hand, with coarse grinding the ash particles are larger and tend to increase the number of fissures. Coarse grinding also leads to an uneven cell structure of the coke because large particles of the individual coals are allowed to carbonize without benefit of mixing. Under any condition, the fineness selected should be held constant.

Bulk Density

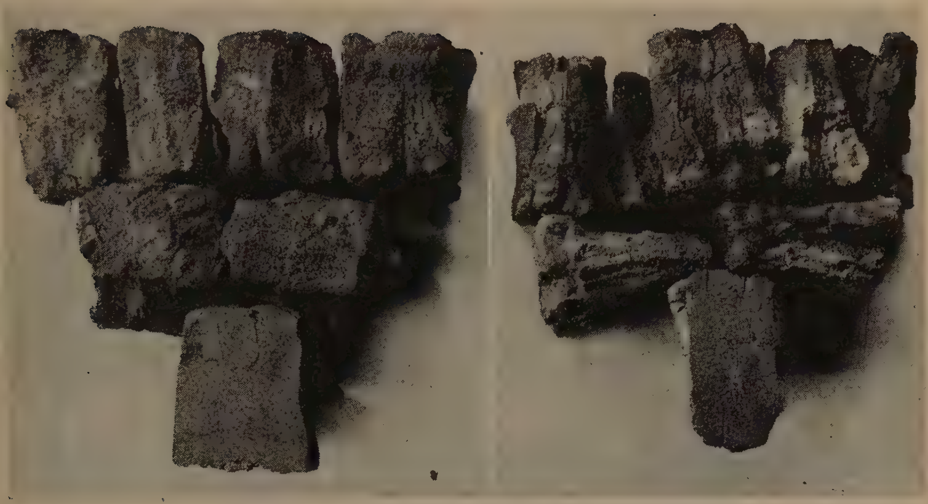
Although bulk density is related to the degree of fineness of the coal, it is also affected by the moisture content. Many times the moisture content of the coal is beyond the operator's control because of weather conditions, or the water content of the coal itself. Whenever possible, however, this factor should be controlled, so that with a standardized degree of fineness the bulk density can be held within practical limits. A recent report[2] on this factor is suggested for study.

When the bulk density of the coal as charged fluctuates, a variable heating rate can be introduced in the carbonization process. The wall temperatures are adjusted to a certain heat input and if the bulk density is higher or less than that normally used, the wall temperature is either insufficient or too high for correct carbonization. Undercarbonized coke is produced in the first instance and overcarbonized coke in the second instance.

When this occurs, it aggravates a condition prevalent in all ovens; namely, different bulk densities because of location and charging characteristics.

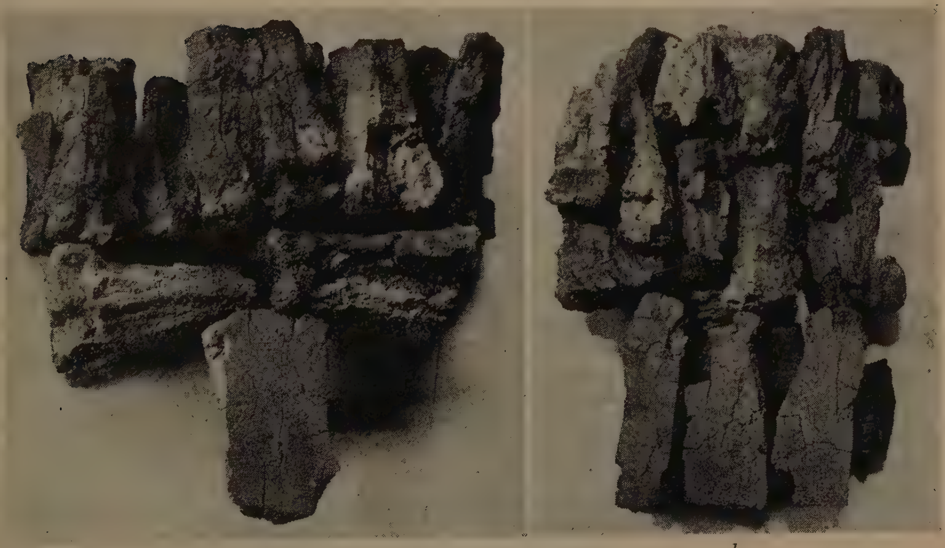
Coking Time or Temperature

In considering coke quality, the temperature of carbonization is either just as



a *b*

FIG. 9.—EFFECT OF COALS ON COKE STRUCTURE.
a. Foundry coke (foundry time). *b.* Domestic coke (foundry time).



a *b*

FIG. 10.—EFFECT OF COKING TIME ON COKE STRUCTURE.
a. Domestic coke (foundry time). *b.* Domestic coke (domestic time).

important as the coal characteristics, or frequently more important. It is not always so regarded but the evidence is irrefutable.

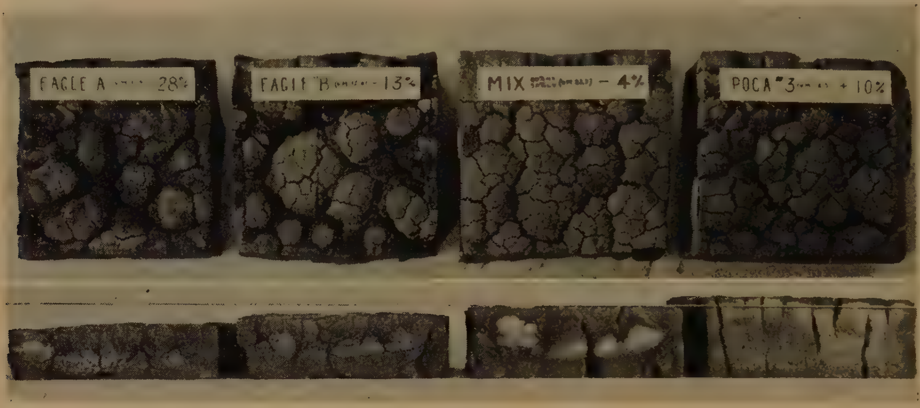


FIG. 11.—COKE SAMPLES FROM TEST OVEN.

In support of this statement, let us quickly review the salient features of the carbonization process. Fig. 6 is an adapted sketch[3] of the progress of carbonization in an oven. With heat applied uniformly to both sides, the heating plane advances from the side walls on both sides toward the center of the oven. This results in parallel planes, first of coke at both walls, then two thin layers of coal in the plastic state, and between these a common layer of coal. Fig. 7 illustrates the actual condition as shown by box tests.[4] In this instance, however, the plastic layer is not visible. Of particular interest is the wedge-shaped growth of the coke at the ends. This, of course, reflects the mobility of heat transfer and the necessity for uniformity of temperature direction, and magnitude.

The rate of heat progression through a coal charge is a direct function of the wall temperature or coking time. For instance, an 18-in. oven at a 15-hr. coking time schedule carbonizes at an average rate of 0.6 in. per hour for half the oven, or 1.2 in. per hour for the full oven, while the same oven at a 30-hr. coking time schedule is carbonizing at an average rate of 0.3 in. per hour for half the oven, or 0.6 in. per hour for the full oven. This means that in the second instance the rate is only half as fast and the temperature differentials are considerably lower. Invariably, the longer the time on the same mix, the larger, blockier and stronger is the coke.

Fig. 8 is a time-temperature curve of an actual oven on a coking time of 27 hr. The couples were mounted as shown approximately 4 ft. into the charge and 5 ft. from the bottom of the oven. The distance from the wall was measured after the test from the couple locations in the coke. The flue temperature was measured directly opposite the couples in the charge. The relative heating rates at the different points in the oven are immediately apparent from the slope of the curves. The starting point of carbonization at each location is also visible. Of particular interest is the fact that all locations show coincident temperature conditions immediately prior to discharging, and that this coke temperature in this particular case is within 150°F. of the average flue temperature. Ryan's[5] work, although done about 20 years ago, is still recommended for study.

The effect of rate of heating is clearly shown in the next two illustrations (Figs. 9 and 10) with the supporting data listed in Table 2.

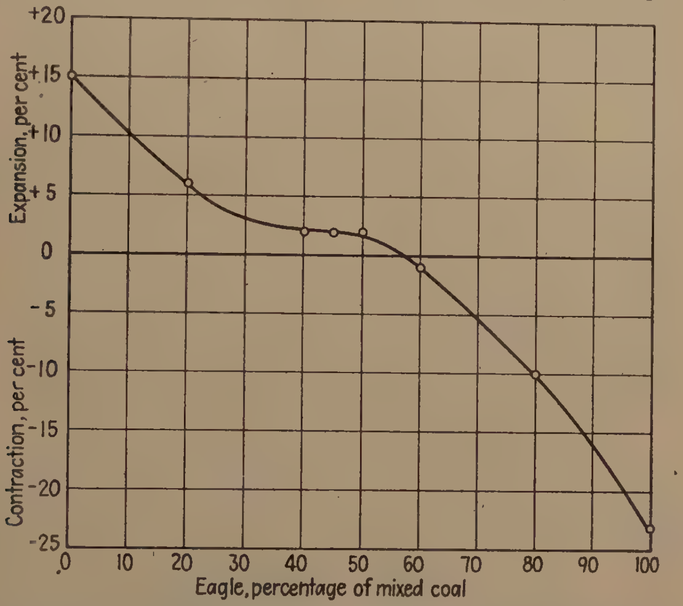


FIG. 12.—BLENDING COALS, EXPANSION AND CONTRACTION.
Bethlehem oven tests on mixtures of high-volatile Eagle with low-volatile Pocahontas No. 3.

In Fig. 10 are shown two cokes made from the same coal mix but at different coking times; namely, 27 hr. and 19 hr. The range of volatile matter in the cokes is approximately 0.6 to 0.9 per cent, so that this factor is of no particular moment. In Fig. 9*a* the foundry coke clearly reflects the benefits of long coking time (low flue temperatures) and good coal blending. The pieces are large, blocky, square fractured, thoroughly carbonized throughout the entire length and with a minimum of internal fissures, both longitudinally and transversely. The domestic blend at the same coking time (*b*) by contrast with (*a*) reflects the influence of the coal mix on the structure. A greater portion of this coke was not full length, reflecting a general weakening of the whole structure. The cut section shows the internal fissure development. The rough external appearance and generally smaller piece is evidence of the deeper and greater development of fissures originating at the hot or cauliflower surface.

TABLE 2.—*R/O Coke Size as Produced*

Mix	Coal, Per Cent			Coking Time, Hr.	Screen Analysis, Per Cent on						Average Size
	H.V.	L.V.	V.M.		6 In.	5 In.	4 In.	3 In.	2 In.	1 In.	
Foundry	50	50	23.1	27	28	51	64	79	91		4.67
Domestic	70	30	26.5	27		17	41	67	89		3.70
Domestic	70	30	26.5	19			4	28	69	90	2.43

Fig. 10*a* shows the same coke as in Fig. 9*b*. Fig. 10*b* is the same coal mixture as in 10*a*, but at a coking time of 19 hr. The contrast is startling. Full-length pieces have almost completely disappeared, the coke is very irregular, heavy internal

fissures have broken the coke into small pieces throughout and the larger pieces left give evidence of an inability to withstand handling without suffering heavy degradation. This shows clearly the influence of coking time or temperature on coke structure. A change of coals would have little effect in counteracting this influence.

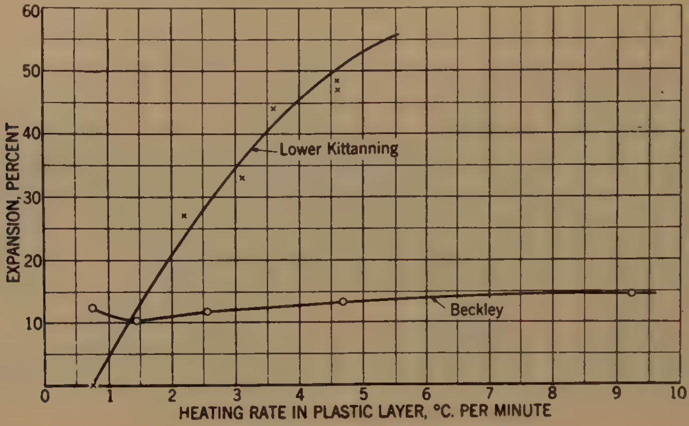


Fig. 13.—Effect of heating on expansion.

Table 2 shows that the size characteristics closely follow the indicated strength or weakness as revealed by the illustration.

An additional test using the 50–50 mix at the 19-hr. time would have been desirable to complete the cross study, but the higher temperatures at this faster time would have caused disastrous expansion, so it was omitted.

The influence of the factors mentioned must be kept in mind if we are to discover the full benefits of any coal or any blend of coals. We dare not disregard or discount the influence of temperature, because when we do we impose a condition of operation that constantly overshadows and nullifies the work of identifying good and bad coals. Moreover, it will considerably delay an improvement in coke quality.

Selection of Coals

The ultimate selection of coal for carbonization is a complex problem because of the many technical and economic factors involved. In considering its technical aspects, we must, for the moment, disregard the economic factors, for a broad scientific investigation must embrace all coals, regardless of location, if we ever hope to discover a fundamental pattern of performance.

It is true, of course, that most individual studies are and will be confined to those coals that are geographically and economically available. The work of the Bureau of Mines is an extremely valuable adjunct to that done by individual investigators in different sections of the country, for it embraces all coals on a national scale. In this manner we who of necessity must confine our studies to particular localities can keep informed on the identification of all types of coal. Often this broad-scale work furnishes the missing link to our own problems, and certainly all of the work must continue to be correlated because of the natural complexity of coal itself.

Of increasing importance too, is the necessity for a coordinated effort of the coke user, the producer and the mine operator because of the connecting and influencing chain of events.

A paper by W. T. Brown[6] is an excellent example of the study directed along the necessary lines. The information it contains is invaluable and should convince even the most skeptical of the breadth of vision needed to accomplish the purpose of better coke through an increased knowledge of the coals used.

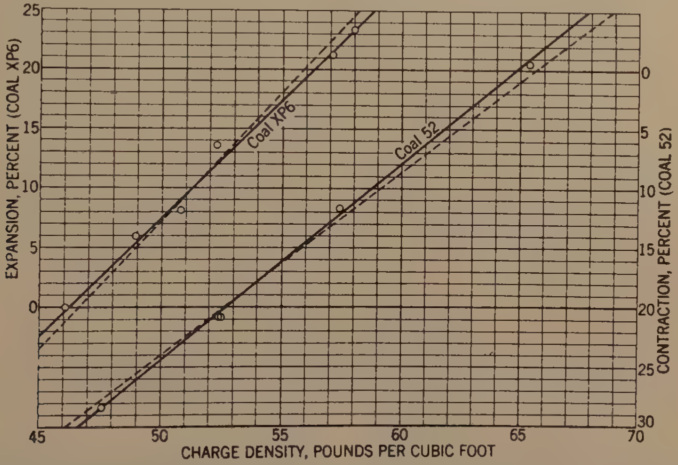


Fig. 14.—Variation of expansion (and contraction) with density of charge.[8]
Solid lines represent actual determined expansion (or contraction); dotted lines represent calculated results correcting to a standard density.[8]

Since it is true that most plant operators are familiar with the general specifications of the coals used, the discussion in the following pages will be confined to fundamentals of coking properties now being revealed or indicated by means of new methods of tests.

Although the methods of test are comparatively new, the factors measured are as old as the industry and have always been known to some extent. For instance, the industry has been well aware of expanding coals and their effect upon the oven structure. Few operators have been able to avoid the experience of "stickers." Coke itself has always given evidence of the fact that the coal had to soften in order to form the cellular structure associated with it, and effect of oxidation on certain coals was always kept in mind when storage in any quantity for any length of time was necessary. What the new tests do, however, which the general knowledge could not do, is to evaluate these factors and assign comparable relationships and indexes.

The apparatus we use for measuring expansion or contraction of coal is a modified Bethlehem tester in which two samples can be run simultaneously. In keeping with the work of others employing similar equipment, we have found that each coal or coal mix has to be subjected to test, for the results cannot be forecast from any general knowledge. It is true that there is a general relationship of high-volatile A-coals contracting and the low-volatile coals expanding, but the degree of either cannot be determined except by actual test. Fig. 11 illustrates this point. The black line represents the original coal level. Eagle A with 30.5 per cent volatile matter, shows a contraction of 28 per cent, while Eagle B, with 32.6 per cent volatile matter shows a contraction of only 13 per cent. In actual practice Eagle B caused "stickers" with a conventional mix of Pocahontas No. 3. The Pocahontas No. 3, with 16.5 per cent volatile matter, showed an expansion of 10 per cent. When Eagle A and the Pocahontas No. 3 were blended 50–50 the mix showed a contraction of 4 per cent.

It is of interest to note that the two coals do not blend in an arithmetical average of their expansion or contraction and this has been found to be true generally.

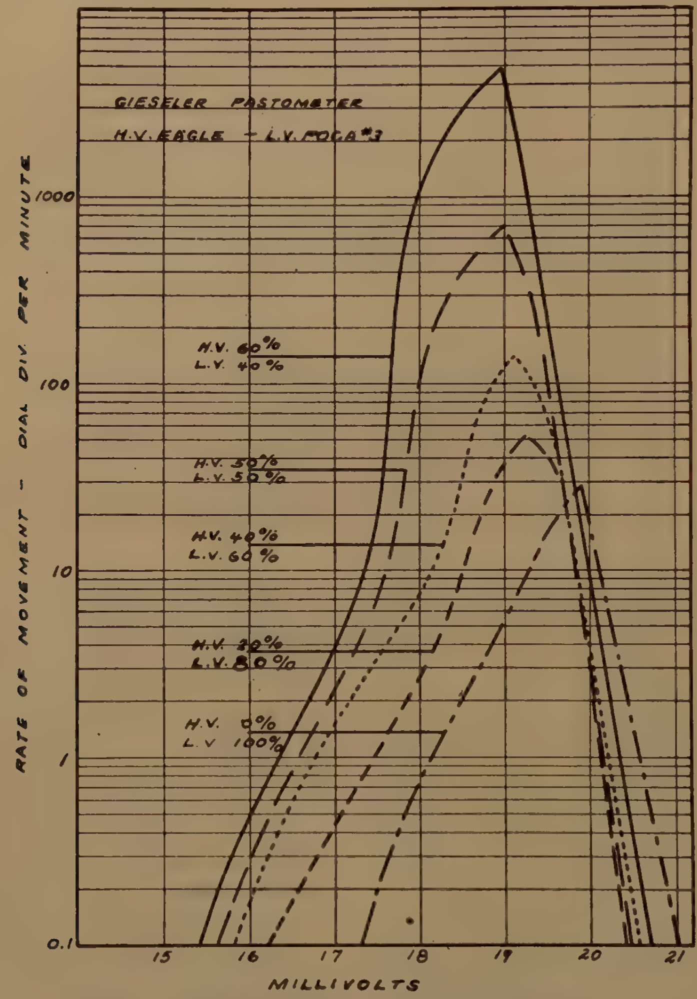


FIG. 15.—GIESELER FLUIDITY CURVES, BLENDING COALS.

An important point shown by these tests is the appreciable difference in contraction of the two high-volatile Eagle coals. Most investigators have placed the blame for damage to oven walls on the low-volatile coals, but this is unfair, for it absolves the high-volatile coals of any responsibility. All coals must be measured and the properties of mixing can then be determined. Arbitrary selection of proportions must be abandoned.

Fig. 12 illustrates a characteristic curve yielding vital information on the blending of two coals. The coals used in this instance were high-volatile Eagle A and a Pocahontas No. 3. This curve again shows that the expansion or contraction is not an average of the individual coals.

These tests, of course, are empirical and because of this they must be conducted under closely regulated and defined conditions. At this stage of their development, too, they must be interpreted in light of their correlation with actual plant practice; that is, in most instances, data from one plant cannot be applied readily to operation at another plant. This does not detract from their value, but simply emphasizes the necessity for a cautious approach or interpretation of their meaning.

To illustrate this point, Fig. 13[7] shows the effect of temperature upon the expansion properties of two coals, and here again there is a striking contrast. The Kittanning shows a precipitous increase of expansion with increase in temperature. This characteristic is not unusual and indicates the necessity for determining this factor at the temperature levels actually used in the particular operation. This factor also is of particular moment in connection with production of blast-furnace coke where fast time and high flue temperatures are generally employed. Reference was made to this factor in connection with Figs. 9 and 10. The Beckley, on the other hand, while showing a measurable increase in expansion with an increase in heating rate, shows considerably less than that exhibited by the Kittanning. We have found this narrow range with high-volatile coals in their contracting characteristics.

Fig. 14[7] shows the effect of bulk density on expansion and contraction. Coal XP6 (Upper Kittanning unwashed) shows a definite and appreciable increase in expansion with an increase in bulk density. Coal 52 (Pittsburgh bed) shows a decreasing contraction with an increase in bulk density. The direction of effect in each instance correlates with actual plant practice, for with certain coals it is necessary to lighten the bulk density by the addition of a greater quantity of water than normally applied in order to prevent "stickers." This factor must be given careful consideration before following the practice of increasing bulk density in order to increase coke yield.

While this test procedure sheds considerable light on the mechanism of expansion or contraction of coals, it is too insensitive to apply to the factor of fluidity. For the determination of this characteristic we employ the Gieseler apparatus slightly modified from its original design. This apparatus is described in the literature.[8,9] This instrument measures the viscosity of a coal during the temperature period when it is plastic. It is of considerable help in interpreting the behavior of different coals. Fig. 15 illustrates one application. Here is shown the change in fluidity of a two-coal mix when the proportions of high-volatile and low-volatile coals are varied. The general characteristic of high-volatile A coals is to be very fluid and low-volatile coals are very viscous. These curves clearly show the change in fluidity that occurs when coals of different characteristics are blended. It shows convincingly

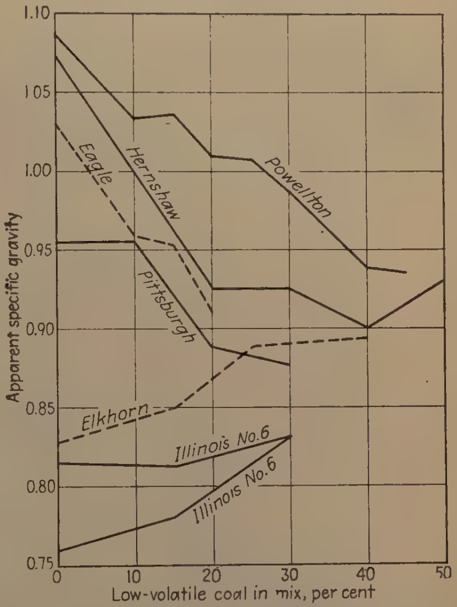


FIG. 16.—EFFECT OF COALS ON APPARENT SPECIFIC GRAVITY OF COKE.[10]

also the necessity for such studies if we are to understand the mechanism of coal carbonization.

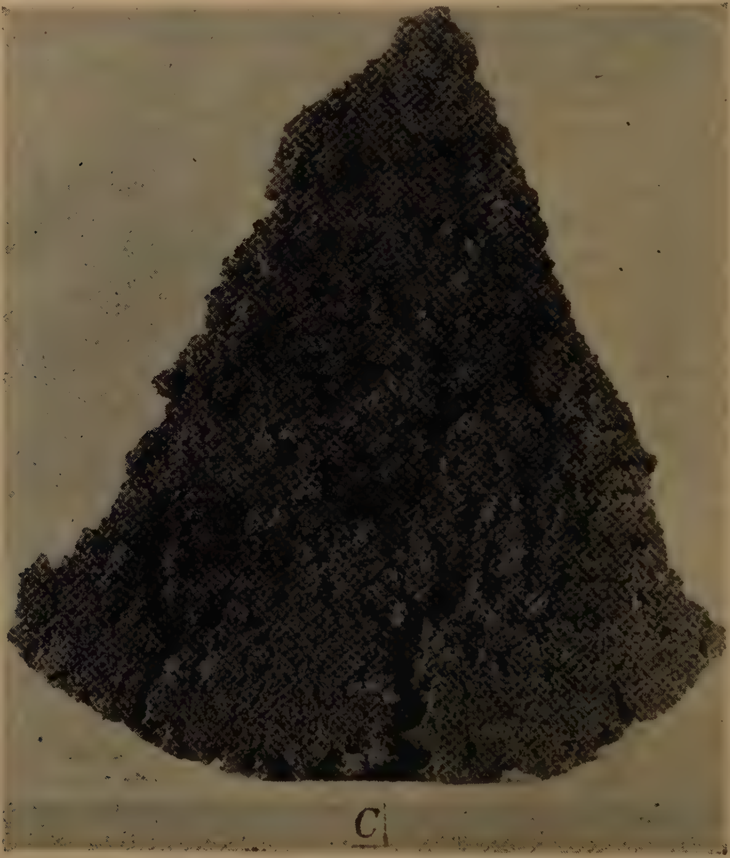


FIG. 17.—COKE FROM OXIDIZED COAL.[13]

In this instrument, highly fluid coals such as high-volatile A froth above the stirrer, permitting an abnormally high rate of rotation. Such speeds are not relative to coals not having this characteristic, therefore the data are not comparable. For this reason we have found it inadvisable to exceed 60 per cent of high-volatile coals (of this class) in this test procedure.

A very interesting effect upon the coke of blending different coals is shown by Doherty[10] (Fig. 16)—the change in specific gravity of the coke with various proportions of high-volatile and low-volatile coals. Fig. 16 shows also the difference in characteristics of high-volatile coals themselves in this respect. This could have been continued to include different low-volatile coals that also show a variation of this factor.

With reference to the operation factors that affect coke structure, the value of the measurements of some of these characteristics must be apparent. The pattern of relationship of these factors is sufficiently defined to warrant greater application for study in order not only to discover wherein these factors are generally applicable but also to discover when they deviate and why.

That these principles of behavior have a very practical application is shown in the methods the by-product coke industry has used from time to time to overcome some of the differences observed in coal. An interesting case in point is the use of additional water to lighten bulk density, and another is in connection with oxidized coals. Most plant operators have had the sad experience of producing some sorry looking coke by the use of old-stock coal and undoubtedly this same coke was sufficiently inferior to cause poor blast-furnace operation.

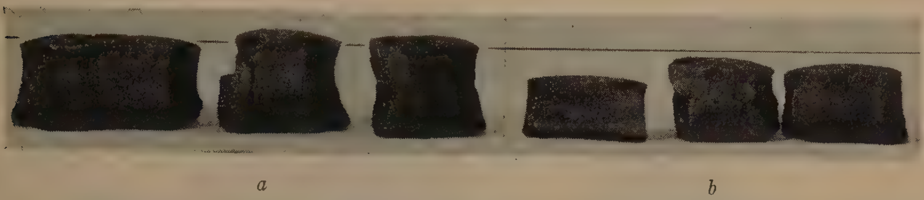


FIG. 18.—MINE SAMPLES OF LOW-VOLATILE COAL.
a. Pure coal section. *b.* Bone section.

Schmidt's[11,12] work on the oxidizability of coals is recommended for study, for here again is a measurement of a factor impeding progress and periodically causing worry.

The principal published work on this factor is that by the United States Bureau of Mines and its work can best be quoted in this regard. A recent publication[13] states: "Oxidation of coal normally affects the yields and properties of most carbonization products, but coals differ greatly in the degree of oxidation necessary to produce significant effects. Decrease in the yield of tar is a sensitive index of the extent of oxidation." An example is given in Table 3.

TABLE 3.—*Oxidation of Thick Freeport Coal Carbonized at 800°C.*

Degree of Oxidation	Yield Tar, Gal per Ton
Unoxidized	16.7
4.0 days	14.5
9.1 days	12.2
13.0 days	11.5
22.1 days	10.7

The effect upon the coke is startling, as shown in Fig. 17. The reduction in fluidity in this instance is reflected in the Gieseler data, in which no definite fluid temperatures were reached.

A number of years ago we stocked several coals and recovered some each month, blending them with each other and with opposite fresh coals to measure the effect of oxidation upon them. A low-volatile coal showed complete loss of fluidity in three months, with a very adverse effect upon the coke, and the high-volatile coal showed no detectable effect in nine months. The greater sensitivity of the low-volatile coals to oxidation has been confirmed several times since these tests and bears out the general findings of the Bureau of Mines. This work is extremely important and it is unfortunate that it has been so long neglected.

Here again is a practical application of a knowledge of these principles. For many operators know and use the practice of higher temperatures when encountering badly oxidized coal that recovers some fluidity lost to oxidation. This, of course, cannot be employed universally, for some coals just will not respond. It has a limited application also when the ovens are already operating at maximum temperatures. The point of interest, however, is the fact that it does fit the pattern of identification of these characteristics with actual plant practice.

Inherent Variables in Coal

The mechanical cleaning and sizing of coal at the mines has progressed noticeably the last few years, but these efforts in general are directed to needs quite different from the consideration of coking properties. It is true that in many instances these improvements have produced better coal for coke, but the immediate factor of distinguishing and isolating carbonizing differences in coal has not been a major objective. This is not the fault of the coal operators; rather, the blame rests with the by-product coke industry for not acquainting them with the information they need in order to help.

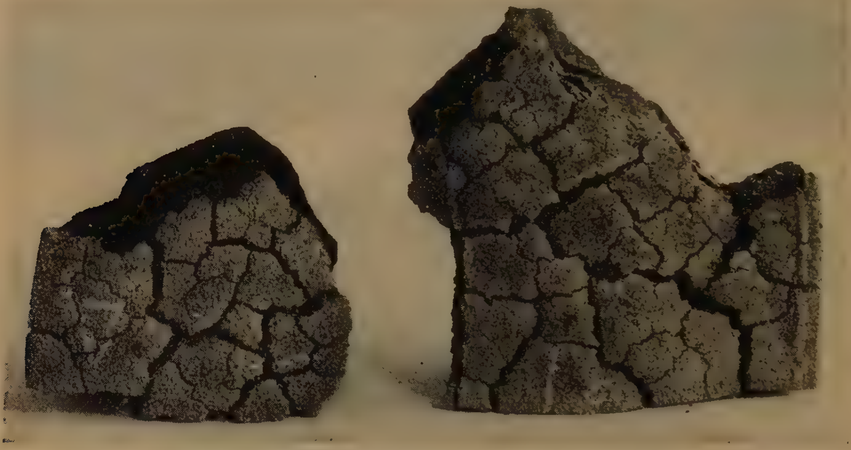


FIG. 19.—MINE SAMPLES OF LOW-VOLATILE COAL. *a.* Pure coal oxidized. *b.* Pure coal unoxidized.

As the identification progressed of the critical factors in carbonization of coal, the need for extending the work directly into the mines became increasingly greater. This applies not only to determination of the characteristics of the coal as it exists in nature but also to the problem of mining preparation.

Brown has discussed some of these factors and Doherty has presented an excellent review of the subject from the coal producers' standpoint. Considerably more work, however, in this aspect of the problem is needed, because many of the variables mentioned and discussed are the result of conditions at this stage of coal recovery and it is essential to the accomplishment of our objective that corrective measures be initiated at that point. An example of some of the findings in this respect from our Eagle mine are shown in Table 4. The mine was sampled in all its principal working areas; particularly at the different vein thicknesses.

TABLE 4.—*Findings at Eagle Mine*

Sample No.[a]	Coal Thickness, In.	Volatile Matter, Per Cent	Ash, Per Cent	Expansion (+) or Contraction (−)
1	84	30.5	3.8	−27
2	86	30.6	4.5	−31
3	66	29.9	4.5	−32
4	59	30.2	4.5	−29
5	59.5	30.4	4.9	−30
19	75	31.5	4.7	−31
9	92	30.6	3.8	−29
10	55	30.1	4.9	−15
16	20	30.5	1.9	+11
23	11	30.6	1.2	+ 8

[a] Samples 1, 2, 3, 4, 5, 19, 9, 10 are full channels; sample 16, upper bench (20 in.) of sample 9; sample 23, second bench (11 in.) of sample 10.

It is interesting to note the uniformity in all the full channel samples in volatile matter, ash and percentage of contraction. The single exception is the contraction of sample No. 10, which seemed to be an isolated condition. This uniformity does not have to exist, nor can it be considered the general experience. The individual bench samples showed no such uniformity and the section thickness of these benches varied appreciably with the total vein thickness. Most of the various benches showed percentage contractions varying from minus 19 to minus 36, which is of considerable moment. The two bench samples, No. 16 and No. 23, gave actual expansions of plus 11 and plus 8, respectively, and this characteristic is extremely important. This deviation of contracting tendencies from the normal behavior of coals of this class can be the cause of serious difficulty in the manner that was mentioned earlier. It is for this reason, too, that the full responsibility for oven-wall damage should not arbitrarily be assigned to the low-volatile coals.

It is now evident that volatile matter content of coal is not a reliable guide or index of its expanding or contracting properties, and this fact alone should conclusively recommend other test procedures for better assessment of characteristics.

Another survey made in connection with a Pocahontas mine is shown, in part, in Table 5. These samples were taken in different working sections of the mine and at widely scattered points. Layer I is the pure coal; bottom benches constituting approximately 65 per cent of the total bed. Layer II is the upper or bone benches, containing a thin layer of coal that was left in the samples. Samples 1 and 2 were somewhat purer than the others and the differences are noticeably greater. The general difference between the upper and lower benches is striking, of course, and yet this is what must be expected.

TABLE 5.—*Survey on Pocahontas Coal*

Sample No.	Layer I			Layer II		
	Volatile Matter, Per Cent	Ash, Per Cent	Expansion	Volatile Matter, Per Cent	Ash, Per Cent	Contraction
1	16.5	5.6	+11	14.1	23.8	−26
2	16.3	5.2	+12	14.1	20.2	−24
3	16.6	7.7	+16	14.5	17.4	−10
4	15.6	6.8	+21	14.9	13.1	− 3.5
5	16.1	3.1	+16	14.6	15.5	− 9
6	15.0	5.6	+22	14.2	13.8	− 3
7	15.7	2.9	+17	14.3	14.6	− 7.5

A few of the samples (coked) from our test oven are shown in Fig. 18. Those on the left are from the pure coal, all showing expansion; those on the right are from the bone benches, all showing contraction. Several of the samples showed appreciable oxidation (Fig. 19). The good coal (right) shows a smooth, normal, well developed cauliflower structure, while the oxidized coal (left) shows the mealy, pebbly surface usually associated with coal of this character.

It has been fairly well established that coal in the mines often shows considerable variation in ash and sulphur, and a recent paper by Toenges and others[14] is a good example of such identification. It is extremely encouraging to learn that attempts are being made to overcome some of these variables in coal by intelligent blending preparation right at their source.

Such studies as those referred to in the preceding pages must be incorporated in the identification of the carbonizing characteristics of coal, for by their isolation at the mine we can materially aid our own work at the ovens in blending and other corrective measures. Without their identification beforehand, we can only continue blindly trying to correct against variables that constantly change in magnitude.

CONCLUSION

It is a difficult task to discuss intelligibly such a complex subject so briefly and in this paper an attempt has been made to high-light the factors we have found to exert the most significant influences on coke structure. We have discovered no quick or simple means for the certain identification of good or bad coals. As with all operators, we lean heavily on our acquired experience initially and proceeding from there we laboriously conduct these identifying tests. We are not so fortunate as always to find the answers we seek, but whatever the results obtained, we feel some contribution has been made.

In spite of the many disappointments and the blind avenues of attack, we are encouraged to believe that a concerted effort by the coking industry along the lines indicated will go a long way to clear the haze from this problem.

The need for a greater effort by all plants is immediate and pressing. We are fully conscious of the depletion of supplies of the better coking coals, occurring at a time too when an improvement in product is constantly demanded because of more rigid specifications imposed in the consuming industries. The answer lies, of course, in intelligent coal selection and blending coincident with an adjustment of operating conditions that will preserve the good features in those coals.

ACKNOWLEDGMENT

The writer gratefully acknowledges the valuable assistance of H. Stuart Auvil, Chief Chemist of the Citizens Gas and Coke Utility, in the work reviewed, and the encouragement of Thomas L. Kemp, General Manager, whose constant desire for better coke gave impetus to many of the studies described.

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A Carbonization Pressure Gauge

By M. A. Mayers,* Junior Member A.I.M.E. and J. A. Thompson*

In recent years, the problem of damage to coke-oven walls by expanding coal charges undergoing carbonization has engaged great attention on the part of research workers in this field, and has led to the development of many devices for estimating these pressures.[1–14] Only in Russell's method[11] and in the Koppers ovens[7] are conditions quite similar to those in commercial ovens. Tests even in these ovens are empirical because no information has been published to permit an evaluation of "scale effects." Owing to the relatively large wall area in these test ovens, the average wall pressure determined may be lower than that existing in localized areas. This factor may be compensated for in part by the testing of coal charged into the test oven at a bulk density that is as high as the highest that is believed to exist in commercial ovens.

Both of these lacks might be filled by the development of an adequate gauge for the direct measurement of wall pressure exerted by the charge in commercial coke ovens in normal operation. Previous attempts to elaborate such instruments[7,15] have resulted in units which, because they had to be water-cooled, probably produced changes in the conditions of carbonization, and which had to be removed from the oven before carbonization was complete, to avoid their destruction. Late in 1941 the Coal Research Laboratory undertook to develop such a gauge, in which gas pressure was balanced against the charge pressure and which thus would not require water cooling, on a suggestion made in 1937 by one of the present authors and William B. Warren, a former member of the Laboratory staff. The gauge heads themselves were to be simple and cheap enough so that they might be discarded after each run if seriously damaged. This would permit them to be left in place throughout the carbonizing period, when they could be pushed out with the coke.

The basis of the design, of which the latest exemplification is shown in Fig. 1, was that as the internal gas pressure within a flexible metal envelope was increased, the flexible wall would move away from a stop when the gas pressure just overbalanced the charge pressure. If the stop were an insulated electrical contact, the point of balance could be indicated by the breaking of an electric circuit. It was anticipated that, while the instrument must operate at high temperature, the presence of a strongly reducing atmosphere throughout the carbonizing period would so reduce the danger of scaling that mild steel could be used for the gauge. This expectation has been justified. Danger of attack on silica oven walls by the steel gauge body can be eliminated by cementing asbestos paper or an alundum or silica plate to the back of the gauge body.

The gauge head itself has a relatively heavy mild-steel body, *A*, into which standard ¼-in. pipe *B* is screwed to conduct gas and electrical leads. The body is closed by a single bellows *C*, consisting of two annuli of 0.011-in. (0.28-mm.) cold-rolled shim-steel spot-welded together at their outside edges. The lower one is

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[1] References are at the end of the paper.

spot-welded to the body, while the upper one is spot-welded to a 1/16-in. (1.6-mm.) mild-steel plate *D*. The latter has a central spot smoothed by grinding for electrical contact, and carries three 1/16-in. (1.59-mm.) diameter pins to hold the outside skirt *E*, made of ⅛-in. (3.2-mm.) plate and a ⅞-in. (22.2-mm.) length of 3½-in. o.d. tubing. The electric circuit consists of a Nichrome V-wire led into the head through a four-hole porcelain insulating tube, which also carries a Chromel-Alumel thermocouple; the stainless-steel contact button *F*, mounted in the central insulator *G*, and the body and connecting pipe of the gauge. Stainless steel was used for the contact button, since it was found that it took on just enough of a coating in the tank nitrogen atmosphere within the gauge to prevent welding to the closure plate without markedly increasing the electrical resistance.

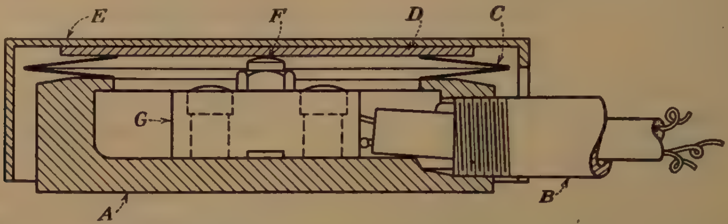

Fig. 1.—Carbonization-pressure gauge head.
A, steel body; *B*, pipe connection; *C*, bellows; *D*, closure plate; *E*, skirt; *F*, stainless-steel contact button; *G*, Lavite insulator.

The central insulators were made of grade A Lavite (American Lava Corporation, Chattanooga, Tenn.) machined and then burned out at 1000°C. for 3 hr. In order to maintain high electrical resistance between the contact and body, the base of the insulator was relieved except around the bolt holes. It was essential that the gauge head and connecting pipe exposed to high temperature be scrupulously clean and free of oil before final assembly, since organic material was found to distill into and carbonize in the pores of the insulator or porcelain protecting tube, shorting the contact-indicating circuit at high temperatures. To clean the connecting pipe, it was heated with a welding torch to burn out the inside varnish layer, and then pickled in a 10 per cent sulphuric acid solution. Pipe connections were lubricated with flake graphite plasticized with a very small amount of light lubricating oil instead of with pipe compound.

When these gauges were tested in the calibrating runs described below, it was found that their indications were erratic during the first hour or so of their lives. This erratic behavior gradually died out and did not reappear when the gauges were cooled down and reheated between calibrating runs, suggesting the existence of residual strains in the diaphragms after welding. When these were eliminated by annealing, the deviations disappeared also. The annealing was accomplished by heating the gauges to 1000° for 2 hr. under negligible load in a box containing a pack-carburizing material through which a slow stream of nitrogen was passed. During the annealing the gauges were kept approximately balanced by connecting them to a manifold supplied with tank nitrogen at slightly above atmospheric pressure.

The balancing gas pressure was supplied to the gauge from standard compressed-nitrogen tanks through a standard reducing valve, and a ballast chamber and pressure-control relay shown in Figs. 2 and 3. It was read on a mercury manometer connected to the ballast chamber. The relay actuates a solenoid-operated valve, so that when the gas pressure within the

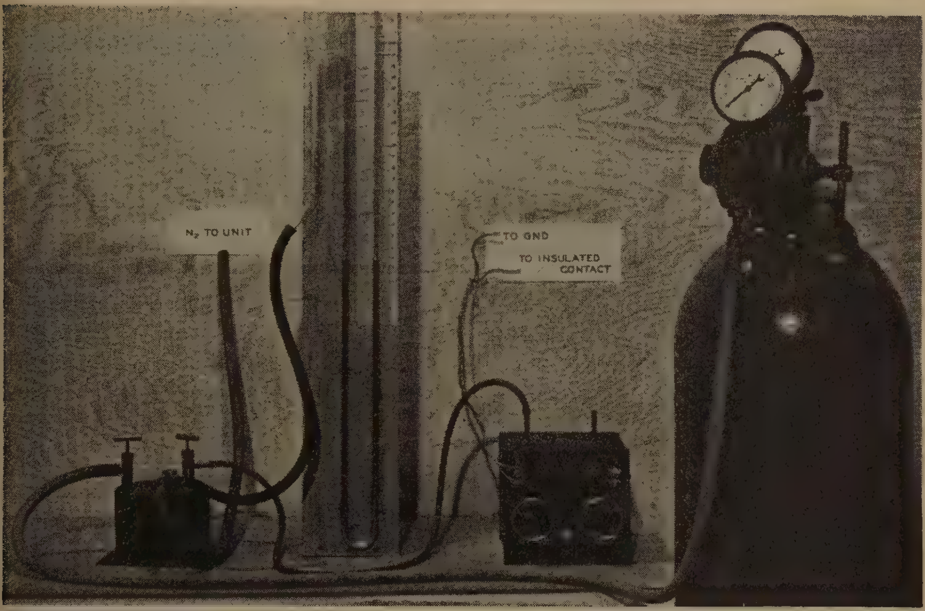

FIG. 2.—BALLAST CHAMBER AND CONTROL RELAY CONNECTED TO GAUGE AND MERCURY MANOMETER.

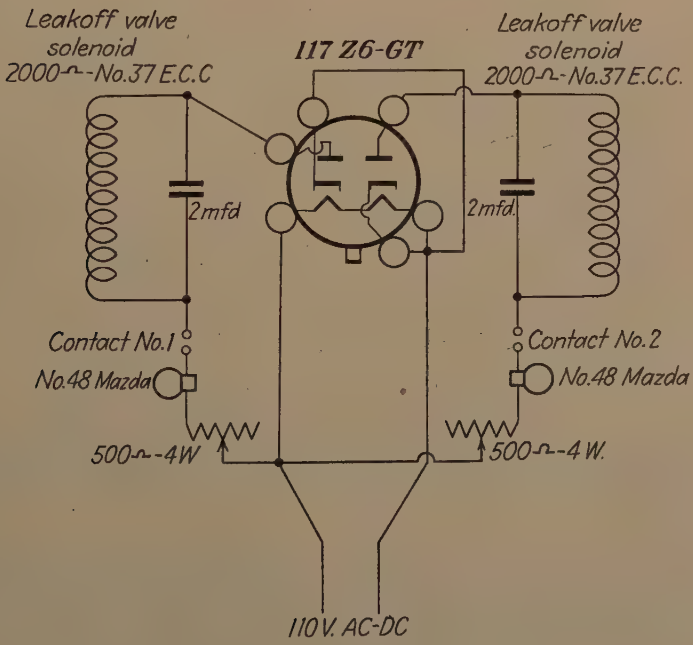

FIG. 3.—WIRING DIAGRAM OF CONTROL RELAY.

gauge is greater than the external load applied by the coking charge, causing an open contact circuit, the solenoid-operated valve is open, so that the gas pressure in the system is reduced. When the gas pressure is less than the external load, so that the closure plate rests against the stainless-steel contact and the contact circuit is closed, current flows to the solenoid to close the leakoff valve and the gas pressure is raised. Thus, the gauge cover is constantly moved back and forth between the open-circuit and closed-circuit positions; under operating conditions a fluctuation in gas pressure of less than 0.2 in. of mercury column (5 mm. Hg) is sufficient. The speed of the cycle can be controlled by manipulation of the needle valves on the ballast chamber, and is usually arranged so that a single cycle occupies about 2 sec. under steady load conditions.

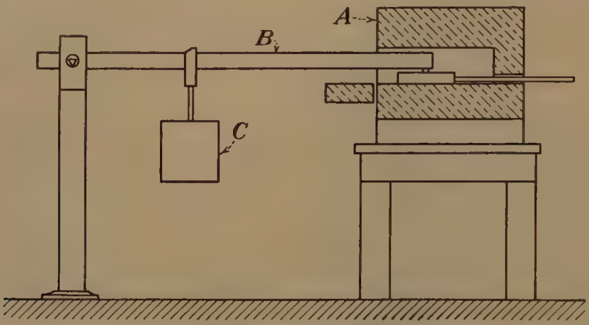

FIG. 4.—LAYOUT OF CALIBRATING DEVICE.
A, muffle furnace; *B*, scale beam; *C*, movable weights.

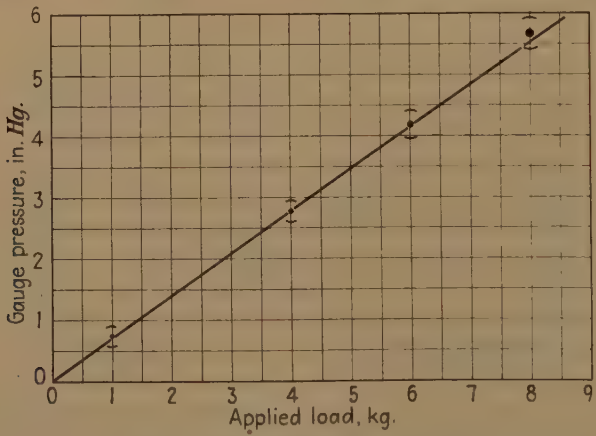

FIG. 5.—RESULTS OF CALIBRATIONS.
The dots are drawn at the average of the observed points with radii equal to the standard deviation associated with the observational precision. The arcs above and below each point have radii equal to the standard deviation associated with the reproducibility of the gauge heads and include the effect of varying temperature.

The gauges were calibrated in a muffle furnace and were loaded by an improvised scale beam projecting into the muffle, with movable weights for applying the loading force, as shown in Fig. 4. The loading scale was calibrated against a standard beam balance set in the position occupied by the muffle furnace. Calculation indicated that the elongation of the scale beam when the muffle was operated at high temperature would introduce a negligible error.

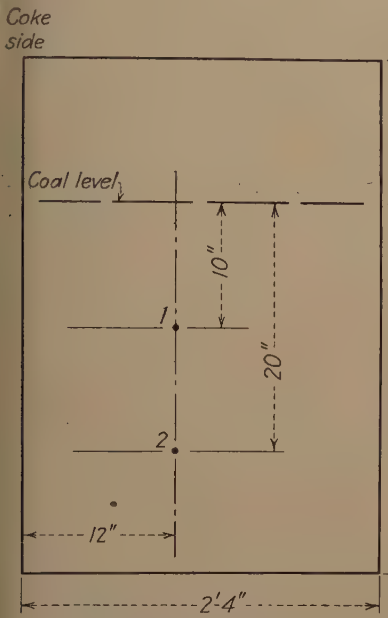

FIG. 6.—ELEVATION OF STATIONARY HOT FACE OF MOVABLE-WALL OVEN.

Point 1, location of upper gauge; point 2, location of lower gauge.

It was found that the gauges had to be carefully protected against oxidizing atmospheres to prevent scaling of the bellows during calibration. The final practice was completely to enclose the gauges in short-metal boxes, closed except for a small opening in the top through which the loading pressure was applied. Each box was half filled with a pack-carburizing material; nitrogen was admitted to it in a slow stream. At high temperature a pale blue flame could be seen burning at the opening around the end of the loading beam. With this arrangement five calibrating runs were made on three different gauge heads, with the results shown in Fig. 5. A statistical analysis of these data showed that all gauges had the same calibration within the limits of error; that the pressure at zero external load is 0.01 in. Hg; and that the slope of the line of Fig. 5 is 0.71 ± 0.002 in. Hg per kilogram load. This is equivalent to 3.1 ± 0.2 lb. per in. Hg in total load on the gauge cover; and if this load represents a uniformly applied pressure, the external pressure is 0.31 ± 0.02 lb. per sq. in. per in. Hg internal gas pressure. Repeated runs on the same gauge heads showed that the calibration was not significantly altered during an exposure of 18 hr. to temperatures above 950°C.

FIG. 7.—TEST SETUP IN MOVABLE-WALL OVEN.

The calibrations described in the foregoing paragraphs indicated that these gauges were reliable enough to secure the desired information if they behaved in the same way in the coke oven as they did on the calibrating stand. In order to gain some operating experience with them under relatively controlled conditions, the Koppers Company, through the kindness of C. C. Russell, made its research department's movable-wall expansion oven[11] available to test the gauges. Three gauges were prepared for testing in this oven; one was installed at point 1, Fig. 6, on the first day's trial, and two were installed at points 1 and 2 on the second day. The gauge at point 1 in the second day's run developed a leak at the threaded pipe connection, so that no observations were taken with it. The source of this trouble has been removed by tapping the pipe thread in the gauge body before counterboring the body. Fig. 7 shows the pipe connections extending out of the leveling door during the second day's trial. Fig. 8 shows the gauge heads in place against the oven wall after the coke had been carefully raked away from them.

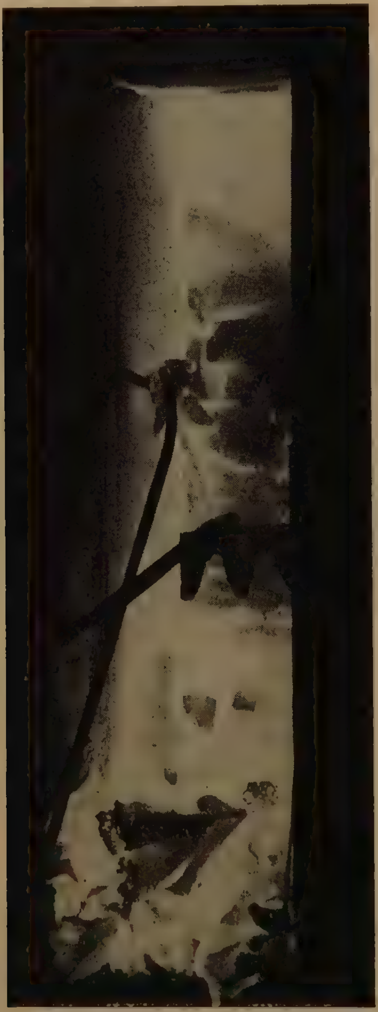

FIG. 8.—POSITIONS OF GAUGE HEADS IN MOVABLE-WALL OVEN ON COMPLETION OF TESTS.

In both runs in the movable-wall oven the same coal mix was used at the same nominal heating rate, although on the second day flue temperatures and rate of heating of the charge were slightly lower than on the first. Graphical logs of the pressure-gauge readings are shown in Figs. 9 and 10, with the curves for average expansion pressure obtained in the movable-wall oven, reproduced through the courtesy of C. C. Russell with the permission of the Koppers Company. Other pertinent data of the two runs are included in Table 1.

It will be observed that the Carnegie Research Laboratory's pressure gauges gave no indication of pressure until the temperature observed at the gauge rose above 900°C. Apparently this effect was due to plastic coal trapped between the body of the gauge and its protecting skirt. Until this coal was carbonized and had shrunk away from the skirt, the gauge was not free to function. Probably this could be avoided by enclosing the entire gauge head in asbestos paper before placing it in the oven. On the other hand, the delay in indication would be less serious in a commercial oven, since the oven wall probably rises to 900° faster than in the experimental oven; moreover, high pressures are observed only relatively late in the coking cycle, so the early stages are not of great interest.

Observations were made on the C.R.L. gauges every 2 min. after their indications started, except for the first 35 min. of the first run, when 5-min. readings were taken. It is evident that the C.R.L. gauges showed much greater detail than did the movable-wall oven; that this detail is significant is suggested by the fact that the same general course was followed by the pressure in both runs. While equivalent points were reached 10 min. later in the second run, superposi-

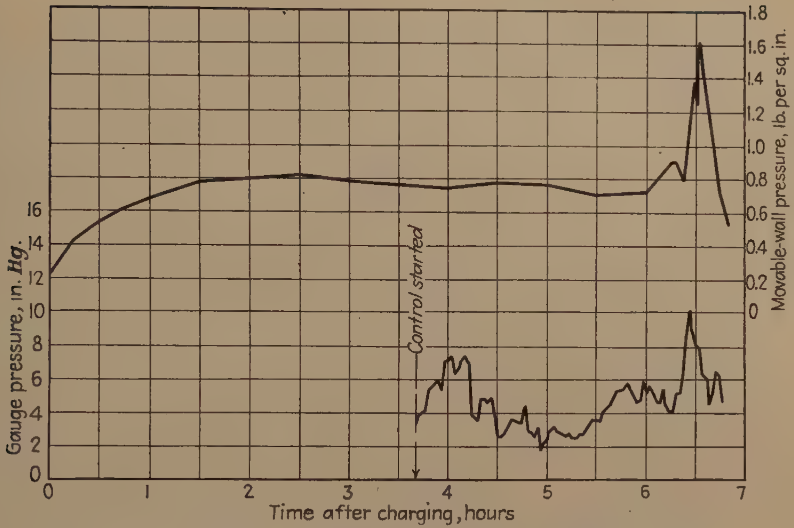

FIG. 9.—GRAPHICAL LOGS OF CARBONIZATION PRESSURES, RUN 1.
Upper part, movable-wall average pressure.
Lower part, C.R.L. gauge pressure.

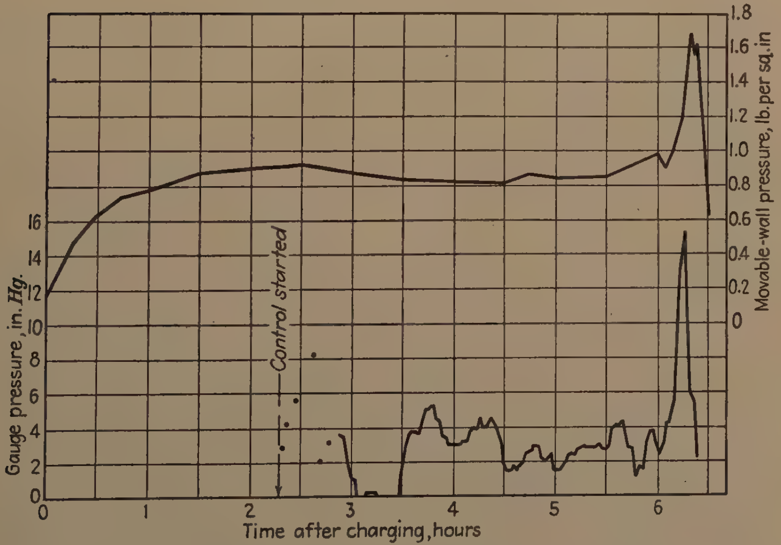

FIG. 10.—GRAPHICAL LOGS OF CARBONIZATION PRESSURES, RUN 2.
Upper part, movable-wall average pressure.
Lower part, C.R.L. gauge pressure.

tion of the curves of Figs. 9 and 10, allowing for this difference in time, shows fair general agreement except for the magnitude of the final peak pressure.

The peak pressures recorded close to the end of the runs were quite different for the C.R.L. gauges, although the average pressures observed on the movable wall were in close agreement. It may also be noted that the C.R.L. gauge peaks occurred 4 and 7 min. before the movable-wall peak in runs 1 and 2, respectively. Expressed as a uniform pressure exerted on the full area of the gauge head, the C.R.L. gauge peaks were 4.75 lb. per sq. in., or 2.82 times the movable-wall peak; and 3.2 lb. per sq. in., or 1.96 times the movable-wall peak, in runs 1 and 2, respectively. The differences between the C.R.L. gauge peaks in the two runs may be associated with the difference in position of the gauges in the two runs, or it may have been caused by the slight difference in the heating rates of the runs; or partly by both of these factors.

The differences between the peak indications of the C.R.L. gauges and the movable-wall oven raise certain other questions. The higher peak pressure indicated by the C.R.L. gauges could be accounted for if these gauges balanced the load on an area greater than that of the gauges themselves. This might be true, because the rigidity of the coke near the hot wall is not negligible. The other possible explanation of the discrepancy allows for the observed difference in time of occurrence of the peaks on the C.R.L. gauge and on the movable wall. Small portions of the oven charge might be heated at different enough rates, depending on varying factors of heat transfer to flue walls, heat transmission through flues, variations in bulk density, size consist, or segregation of the charge resulting in differences in rate of advance of the coking zone through the charge, so that the times at which maximum pressures would occur, as well as the magnitudes of those maxima, might differ from point to point throughout the oven section. Thus, the movable wall would observe only the sum of all these effects; if the individual maxima were sufficiently localized and far enough out of phase, the peak average pressure might be expected only after all the individual peaks had occurred, or at the time of the latest ones, and would be lower than the peak pressure at any point. That this may be the mechanism of the development of destructive pressures in commercial ovens is supported by the known fact that damage from this source is likely to be localized. It suggests that devices indicating average pressures over relatively large areas, such as those now in use, may be unreliable indicators of the maximum local pressures attained in oven practice, and that some of the difficulties in the correlation of the results of tests with such devices with one another and with commercial practice may be the result of that defect.

SUMMARY

A progress report of a continuing investigation of the mechanism of carbonization of coal has been presented, describing a gauge prepared for the observation of the mechanical pressure against a point of the wall of a slot-type coke oven by the charge undergoing carbonization.

It has been shown that the gauge head has a life at high temperatures considerably greater than the time of coking in commercial ovens, and that it maintains its calibration, which can be reproduced in different units within the required limits by taking suitable precautions in making them, throughout this life.

Tests of units of this type have been made in an experimental movable-wall oven. The pressures observed have been several times as great as the average pressures observed against the movable wall. This may be due to transmission of a load corresponding to an area greater

TABLE 1.—*Tests in Movable-wall Oven*

Data	Run No. 1	Run No. 2
Bulk density of coal	51.3	52.0
Coal mixture......	80 per cent Eagle, 20 per cent Pocahontas	80 per cent Eagle, 20 per cent Pocahontas
Moisture, per cent..	1.9	
Fineness, per cent:		
Through ⅛ in.	82.6	81.6
On ⅛ in., through ¼ in.	15.6	17.3

Time		Temperatures					Pressure on Movable Wall, Lb. per Sq. In.	Temperatures					Pressure on Movable Wall, Lb. per Sq. In.
				Coke						Coke			
				Distance from Wall						Distance from Wall			
Hr.	Min.	C.R.L. Gauge, Deg. C.	Flues, Deg. F.	0	Point 1	Point 2		C.R.L. Gauge, Deg. C.	Flues, Deg. F.	0	Point 1	Point 2	
0	0		2568				0.16		2576				0.20
	30	802					0.63						0.53
1	0	834		884	678	36	0.78	698		820	577	33	0.67
1	30	866					0.87	753					0.77
2	0	896			920	44	0.90	787		886	813	45	0.79
2	30	916					0.92	784					0.81
3	0	934		959	832	86	0.87	823		911	869	89	0.78
3	30			969	847	89	0.83	896					
4	0		2585				0.82	931	2577	931	904		0.74
4	30	972		977	871	89	0.81	956					0.77
5	0	982					0.84	966		949	931		0.76
5	30	993		998	898		0.85	985					0.70
6	0	1004					0.98	987	2570	962	949		0.72
6	5						0.90						
6	10						1.02						
6	15						1.20						0.89
6	18												0.91
6	20						1.68						
6	22						1.56						
6	23						1.61						0.79
6	30		2577				0.63						
6	31												1.25
6	32												1.55
6	33												1.63
6	35												1.46
6	45							1004					0.73
6	50									969	962		0.54

than that of the gauge to the gauge head, because of the rigidity of the charge of coke, or it may be due to differences in the time of occurrence of the maximum pressures at different points in the wall, so that the movable-wall records an average peak only after most of the individual peaks have occurred, giving a low indication of the maximum localized pressures.

ACKNOWLEDGMENT

The assistance of the Koppers Company, and especially of Mr. C. C. Russell, in securing this information, and in granting

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Sources of Pressure Occurring during Carbonization of Coal

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(Pittsburgh Meeting, October 1943)

A PRIMARY factor in the selection of coals for making coke at high temperatures is the amount of pressure the coal will exert upon the oven walls when carbonized in modern by-product ovens.[1–3] This factor has become increasingly important in recent years because of demands for increased throughput and higher coke yields. As a result of the increasing necessity for assessing coals in regard to their carbonization pressures, more and more research work is being directed toward developing laboratory-scale tests for predicting carbonization pressures and for showing how blends of coking coals may be modified to eliminate pressures liable to damage coke-oven walls.

The specific reasons have never been defined as to why some blends of coking coals develop dangerous carbonization pressures and others do not, therefore specific procedures to eliminate dangerous pressures could not be prescribed. It is true that the major processes occurring during carbonization have been described by many writers,[4] and there is a general agreement on the ideas of the changes that occur as coal is transformed into coke. However, there is little evidence relating the processes occurring during carbonization to the pressure developed and the available evidence is very incomplete and often contradictory. This deficiency arises in part from the fact that the fundamental causes of pressure development have not been defined, at least from experimental data, and in turn this is the result of the lack of satisfactory testing equipment for studying the problem.

Operating experience has shown several ways of reducing or eliminating dangerous pressures,[5] but probably none of these ways is applicable to all blends of coking coals, and such empirical methods often lead to production of inferior coke. They are trial-and-error methods, not designed to remedy specific causes of pressure development.

The number of laboratory-scale testing devices designed to estimate the carbonization pressures that result when coals are carbonized in commercial ovens is probably well over a hundred.[6] Few of these instruments have remained in use for any length of time, either because the predictions were not sufficiently accurate or, more often, because too many exceptions were found. Often testing devices are operated without any attempt at duplication of conditions in commercial ovens and erroneous and contradictory results are frequently obtained. Therefore, it has seemed desirable to study the properties of coal itself and to determine any relationships between the properties of the coal and the carbonization pressures developed.

RELATIONSHIP BETWEEN PROPERTIES OF COAL AND CARBONIZATION PRESSURE

Attempts to determine relationships between various properties of coal and carbonization pressures, for the most part, have been indirect because no satisfactory and reliable laboratory testing device has

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[1] References are at the end of the paper.

been available for determining pressures that would be developed in commercial ovens. The usual procedure has been to conduct analyses of coal properties after coals have been found to be dangerous in commercial use and then attempt to relate the coal properties with the dangerous pressures. Papers have been published to show relationships between carbonization pressures and such properties of coal as volatile matter content, plastic properties, ash content, petrographic composition, swelling characteristics and moisture content.[2,3,5,7–22] Most of these studies have obtained some broad relationships but either exceptions have been found for which no satisfactory explanation could be offered or the results were too general to be applied to specific coals. Thus, one could never be sure that a given coal would behave in a commercial oven in the manner predicted by laboratory tests.

The use of laboratory-scale analyses of coal properties as presented in this paper is believed to be a new approach to the problem of assessing coking coals in respect to their expansion properties. By means of the laboratory tests described, it seems possible to obtain from less than a pound of coal sufficient information to determine whether individual coals or specific blends of coals are safe to use under ordinary coke-oven conditions. These predictions are modified, of course, when unusual conditions are present, such as high bulk density, different heating schedules, unusual moisture content or degree of pulverization of the coal. As yet, little work has been done toward utilizing the methods presented in this paper to predict coke quality, but, from several broad generalizations noted in the course of the work, a possible way to use this information to predict coke quality as well as carbonization pressures is suggested.

The experimental work described is divided into two parts: (1) measurement of the pressures developed during carbonization of a given coal or a blend of coking coals by means of the large movable-wall test oven, and (2) laboratory-scale analyses of the coal or coal blend. The analyses were examined to determine how they could be used to account for the pressure curves obtained from the movable-wall test oven. It has been found that, from the knowledge given by proximate analysis, the plastic properties as measured by the modified Gieseler-type plastometer, the rate of volatile-matter loss measured over the plastic period and on each side of it, and the conditions of the movable-wall oven test, the carbonization-pressure curves could be explained. Approximately two hundred tests in the movable-wall oven have been examined and the laboratory-scale tests that had been made on those coals have been found adequate to explain the pressure development. In a few cases, it has been possible to make satisfactory predictions of pressures for blends of coals that were expected to show quite different pressure development.

Mechanism of the Coking Process

Laboratory study and operating experience have given a general idea of the processes that occur in a coke oven in the transformation of coal to coke. These processes may be outlined briefly in order to give a basis for discussing the pressure curves obtained from the movable-wall test oven. The processes described are, of course, broad generalizations. Many subordinate reactions take place in the course of the transformation of coal to coke, any one of which may be of particular importance in a specific instance. Most of these subordinate reactions are not clearly defined and often are the subject of controversy. Only the broad general processes will be used to describe the carbonization processes, because they are generally accepted by fuel technologists.

The carbonization process may be divided into four parts or phases, with no definite boundary between the parts: (1) charging coal into the oven and the development of the plastic seams, (2) progression of the plastic seams toward the center of the oven, (3) meeting of the plastic seams in the center of the oven, and (4) disappearance of the plastic seams and the transformation of the center portion of the charge into high temperature coke.

In the first phase, coal, usually in a pulverized condition, is dropped into a coke oven, where the temperature may be as high as 1200°C. The coal touching the sides of the oven attains immediately a temperature high enough for rapid decomposition and softening. In a very few minutes the coal touching the walls is transformed into semicoke and the heat transfer has proceeded so that the layer of coal immediately adjoining is then in the plastic state. At this time the temperature gradient per unit distance between the walls and the unchanged coal is very great. After perhaps an hour the process has proceeded far enough so that the temperature gradient extends over a considerably greater thickness of charge and the gradient per unit distance across the plastic layers is much smaller than in the first few minutes. Thus after an hour or so the process has attained a relatively stable form, coke at the oven walls, unchanged coal in the oven center, and two layers or seams of plastic coal separating the coal and the coke.

The second phase of the carbonization process occurs over the greater part of the coking period. After the charge has attained the form described in the first phase, continued application of heat causes the continuous change of coal into plastic coal and plastic coal into coke, the processes proceeding progressively farther from the oven walls. The thickness of the coke layer at the walls increases progressively, the thickness of the unchanged coal remaining in the oven center decreases, and the two layers of plastic coal appear to move toward the center of the oven. In the production of high-temperature coke the plastic layer moves at an average rate of about ½ in. per hour and, since there are two plastic seams, one from each wall, approximately one inch of coke is formed per hour.

The meeting of the two plastic seams marks the third phase of carbonization. When the plastic seams meet, unchanged coal disappears from the oven and only the plastic coal and the coke remain. It has been shown that the rate of heating of the united plastic layers is much more rapid than the rate during the second phase of the coking.[23] This is attributed to the fact that the heat transfer to the now single plastic layer occurs from both sides and there is no heat transfer from the plastic layer to unchanged coal at a lower temperature. Thus, the change in the fluid properties and the rate of decomposition in the plastic layer is more rapid than in the greater part of the coking period.

The plastic layer disappears perhaps an hour or less after the meeting of the two individual layers, leaving only coke in the oven, ranging in temperature from perhaps 500°C. at the oven center to 1100°C. at the oven walls. The only change that occurs after this is the transformation of the lower temperature coke to high-temperature coke, which may be called the fourth phase of the carbonization process. This fourth phase of the process is characterized by continued evolution from the coke of gas rich in hydrogen, and this continues up to the time of pushing the coke.

This general description shows that except for the last part of the coking process the charge in the oven consists of coke at the walls and two layers of plastic coal adjoining and separating the coke from the unchanged coal in the center of the oven. Decomposition of the charge occurs during the entire process, ranging from the evolution of water beginning at about 100°C. in the coal itself to the evolution of tar, oil, and gases during later stages, and ending with the evolution only of gases from the coke. The rate of decomposition, as will be shown later, is generally greatest within the temperature range of the plastic layer. This evolution of volatile matter within the plastic layer, and the resistance of the plastic layer to the escape of the volatile matter produced within it, is generally believed to be the source of pressures developed during the coking process. The experimental work presented in this paper is supporting evidence for this belief.

PRESSURE CURVES OBTAINED WITH MOVABLE-WALL TEST OVEN

The large movable-wall test oven used in this work was described in an earlier paper.[24] Similar test ovens have been used by the U. S. Bureau of Mines[25] and in Germany[3] for measuring carbonization pressures. This large oven appears to be the only satisfactory device for measuring carbonization pressures as they probably occur in commercial ovens. It is possible that some other smaller testing devices will be found satisfactory when the results can be interpreted properly.

This oven may be described briefly as a test oven with a chamber 12 in. wide, 30 in. long and 39 in. high. One wall of the oven is mounted on rollers and is equipped with a lever-weight system for measuring the pressure developed against the wall by the charge during the course of the carbonization. The chamber holds a little more than 8 cu. ft. of coal, or about 450 lb. at normal bulk densities, 49 to 53 per cu. ft., so the apparatus can hardly be classed as a laboratory-scale instrument. The movable wall actually moves only a few thousandths of an inch during tests of coal blends used in commercial ovens, so that tests are conducted under essentially constant volume conditions.

The test procedure calls for dropping the coal through the top of the oven as in commercial practice. Flue temperatures are regulated so that normal coking time is 7 to 8 hr., which is comparable to 15 to 16 hr. in commercial 18-in. ovens. Coke made in this oven compares favorably with coke from commercial ovens in physical characteristics except for the smaller top size and slightly lower apparent specific gravity. The latter may be explained at least in part by the fact that the coke usually is pushed just after the disappearance of the plastic layer and has not been heated so that the entire charge might be called high-temperature coke, as in commercial ovens. Another cause of the lower specific gravity is that the height of the coal in the oven is less than in commercial ovens. The lack of weight over the coal near the sole of the oven appears to lead to somewhat lower gravity for the coke.

The curves obtained by plotting carbonization pressure against coking time may be divided into six types according to several fundamental characteristics of the coal tested. It should be remembered that these are not separate and distinct types with definite limits. Pressure curves that may fall anywhere between any two types are possible. However, the separation of the curves into types according to fundamental characteristics of the coal as reflected in the shape of the curves has aided materially in the identification of specific causes of dangerous pressure development.

Fig. 1 shows examples of each of the types of pressure curves. Curves of actual tests have been used for these examples, to emphasize that fundamental characteristics of the coals tested are reflected in the pressure curves.

Pressure curves of type 1 are characteristic of coals or coal blends for which the factors tending toward pressure development are at minima. The pressure curve begins at 0.1 to 0.2 lb. per sq. in. and does not vary appreciably from this range throughout the test, the starting pressure being the pressure of the weight of the coal against the oven wall. The curve is typical for the coking of many 100 per cent high-volatile coals.

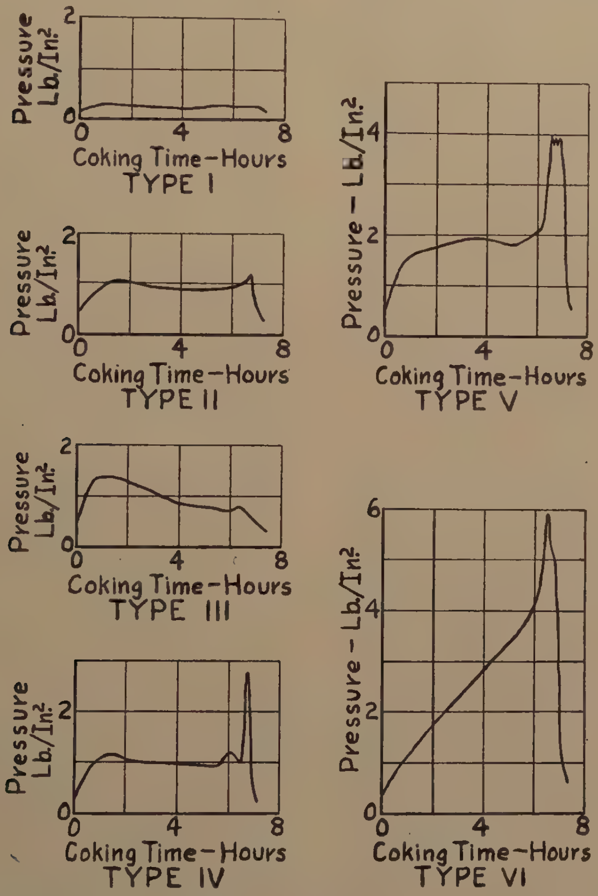

FIG. 1.—PRESSURE CURVES OF TESTS IN MOVABLE-WALL TEST OVEN.

The type II curve is typical for many coal blends used in commercial ovens for the production of blast-furnace, water-gas, and domestic coke. After the initial pressure of a few tenths of a pound per square inch for the weight of the coal against the oven wall, the pressure curve rises rapidly for the first hour or so and then levels off. This initial rise in pressure during the formation of the plastic seams often attains a value as high as 1 to 1.2 lb. per sq. in. For the next 5 hr. the pressure remains nearly constant or decreases slowly from the initial maximum attained in the first hour. When the plastic seams meet, after about 7 hr. of coking, there is a slight increase in pressure, which may amount to several tenths of a pound per square inch, and usually is the highest pressure shown during the test. After this maximum, the pressure declines very rapidly as the plastic layer disappears and the coke remaining shrinks to relieve the pressure on the wall.

Blends of coking coals containing large percentages of so-called high-oxygen coals, coals of high-volatile A rank[26] but approaching the lower rank high-volatile B group, or of high-volatile B rank, give carbonization pressure curves of type III. They are characterized by a very rapid increase in pressure during the first hour, the maximum pressure sometimes exceeding 1.5 lb. per sq. in. During the next 5 hr. the pressure declines fairly rapidly, so that when the plastic seams meet the final maximum or peak pressure is usually less than the initial maximum pressure.

Pressure curves of types I to III are all for coals or blends of coals normally safe to use in commercial ovens. Coal blends giving curves of type II or III may give much higher pressures when charged at higher bulk densities. Types IV to VI are pressure curves for coals or blends of coals that are dangerous to use in commercial ovens at normal bulk densities.

Type IV curves are very similar to type II, the distinguishing feature being the occurrence of a dangerous maximum pressure when the plastic seams meet in the oven center. From the initial pressure the curve rises rapidly during the first hour and attains a maximum value of around 1 lb. per sq. in., then the pressure declines slowly but continuously for the next 5 hr. or so. Up to this time the curve is the same as obtained for blends giving type II curves. When the plastic seams meet in the oven center, near the seventh hour the pressure rises very rapidly in a few minutes, attaining values of 3 or 4 lb. per sq. in., which are above the accepted limit of safety. After this maximum, the pressure decreases very rapidly as the plastic seams disappear and the coke shrinkage relieves the pressure. Coke from coals of this type of pressure development has much the same physical characteristics as coke from coals giving type II curves.

Type V curves show a rapid rise in pressure during the first hour, attaining a value of 1.5 to 2 lb. per sq. in. The curve continues to rise but very slowly or declines somewhat during the next 5 hr. of the coking period. This relatively constant, moderately high pressure continues all during the course of the progress of the plastic layers to the oven center. When the plastic seams meet, a further rapid rise occurs until the plastic coal disappears. The peak pressure at the meeting of the plastic seams usually exceeds 4 lb. per sq. in. After the plastic seam disappears the pressure declines rapidly as coke shrinkage again relieves the pressure. Coal blends giving this type of curve may be of three kinds: (1) those containing large amounts of low-volatile coal and coked at normal bulk densities, (2) blends that normally give pressure curves of type II but coked at unusually high bulk densities, and (3) blends with unusual plastic properties, which will be discussed later.

Charges of 100 per cent low-volatile coal give pressure curves of type VI. Actually the curve used for the example was at an abnormally low bulk density. If this same coal had been tested at a normal bulk density, the pressure curve would have had a much steeper slope and the maximum pressure would have exceeded the limit of the apparatus (9 lb. per sq. in.). In such a case the low-volatile coal exerts sufficient pressure to move the oven wall an appreciable distance. The pressure rises very rapidly from the moment of charging and the rapid rise of pressure continues as long as plastic coal is present in the charge. The rate of increase of pressure accelerates when the plastic seams meet, but when the plastic coal disappears the pressure drops rapidly because of the relief offered by the shrinkage of the coke remaining.

These six types of pressure curves include all of the tests made in the large movable-wall test oven, the number of tests at this writing being approximately 300. Nearly all of these tests have been made at normal bulk densities, 49 to 53 per cu. ft.; that is, the coal charged was of low moisture content and had a pulverization of approximately 80 per cent through a ⅛-in. screen, and the heating schedule was much the same. The few tests at unusual bulk densities also fit into this classification when allowance is made for the unusual conditions. Gieseler plastometer data were obtained at the time of many of the tests.

Laboratory Tests That Explain Pressure Curves from Large Movable-wall Test Oven

The source of pressure arising during the carbonization of coal, as stated earlier in this paper, is believed to be resistance of the plastic coal to the escape of volatile products formed within the plastic seams; therefore laboratory tests to study the pressure development must include measurement of the plastic characteristics of the coal and also estimation of the evolution of volatile matter during the plastic temperature range. Additional information on the degree of shrinkage of the charge, the apparent specific gravity of the coke,[27] and the usual proximate analysis were found useful for further qualification of the development of dangerous pressures.

Plasticity Tests of Coals Made with the Modified Gieseler Plastometer

The Gieseler plastometer has been described in the literature,[28,29] so only a brief description of the instrument will be given here. A few grams of coal is packed around a stirrer with rabble arms in a retort, so that the stirrer will not move at room temperature when a given force acts to turn it. The retort is heated at the rate of 3°C. per min. starting at 300°C., a rate of heating that has been found to be about the average over the plastic range in modern coke ovens.[30] When the coal softens, the force acting to turn the stirrer causes the stirrer to revolve in the coal. The rate of turning of the stirrer is measured on a dial and expressed as "divisions per minute." One complete revolution of the stirrer equals 100 divisions. The course of a test shows that a coal becomes gradually more fluid up to some maximum fluidity and then becomes progressively and rapidly less fluid until the plastic coal "sets" to semicoke and the stirrer no longer moves.

The modified Gieseler plastometer used by Koppers Company's Research Department has been calibrated by using tars and pitches of known viscosities, so that the Gieseler units of "divisions per minute" movement can be expressed in absolute viscometric units, poises.* Thus, a dial movement of 100 divisions per minute indicates that the softened coal system has a consistency of 76,000 or 7.6×10^4 poises.

The usual method of reporting Gieseler plastometer tests[31] is shown in Fig. 2, where the stirrer movement is plotted against temperature. The curve shown is a typical result of a plastometer test on a high-volatile A-rank bituminous coal. For the purposes of this paper it has been found desirable to use a different method of reporting the plastic properties, as shown in Figs. 7–12. The temperature on each side of the plastic range at the fluidity of 50 divisions per minute, or a viscosity of 1.5×10^5 poises, is indicated, the significance of which will be described later. The lines along the temperature scale are extended to intersect the curve for the rate of volatilization, which also may be plotted on the same temperature scale.

Measurement of Rate of Volatilization of Coals

Various authors have reported experimental work on the decomposition and loss of volatile matter of coals[32–49] during the temperature range of the plastic state. Most of this work, however, has been done with little or no attempt to approximate commercial coke-oven conditions. Usually the measurement of volatilization has been restricted to the determination of permanent gas evolution. Since the volume of gas produced during carbonization is much the same for most coking coals, either of low, medium, or high-volatile rank,[50] permanent gas volume is hardly a criterion for differentiating between coals. The composition of the gas, on the other hand, varies greatly. In addition, the tar, water, light-oil yields, etc., vary from coal to coal. It would require a complex setup to measure the volume of the total volatile products being produced at small temperature intervals during the carbonization period. However, the measurement of the rate of loss of total volatile products during carbonization is relatively simple when performed on a weight basis.

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* The term "poise" is the centimeter-gram-second unit used to express the viscosity or flow characteristics of liquids. It expresses the internal friction of a liquid or the resistance of a liquid to motion of its parts. If the tangential force per square centimeter exerted by a layer of fluid upon another layer of the fluid adjacent to the first is one dyne when the first layer is moving at a rate of one centimeter per second, the viscosity of the fluid is one poise. For example, at room temperature water has a viscosity of 0.01 poise while molasses has a viscosity of approximately 1000 poises.

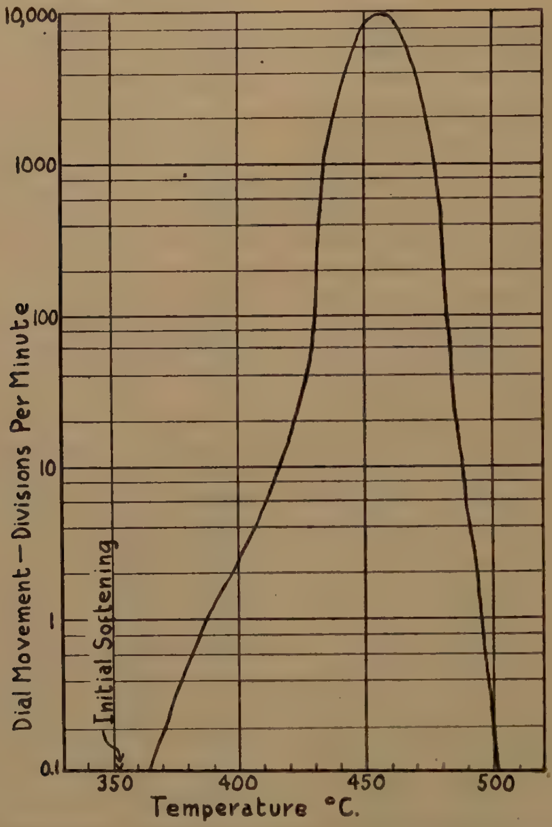

FIG. 2.—GIESELER PLASTOMETER TEST OF TYPICAL BLEND OF COKING COALS.

Measurement of the rate of loss of volatile matter has been carried out on a weight basis and the results have been found to be of considerable use in explaining pressure development for various coals. The setup used is shown in the diagram of Fig. 3. Five grams of coal is placed in a cylindrical steel crucible and the crucible is suspended from one arm of an analytical balance. A thermocouple is placed in the center of the coal sample and the leads are coiled and adjusted to minimize the restriction of movement of the crucible suspended on the balance. The balance scale is calibrated so that the change of location of the pointer on the scale can be reported directly as milligrams change in weight, thereby minimizing adjustment of individual weights on the balance during tests. An oil damper is used to damp the swing of the balance beam. The crucible is placed in an electric furnace and heated to 300°C. in about ten minutes. Thereafter, the heating rate is maintained at 3°C. per minute by an automatic heating-rate controller, the same as for the plasticity tests. A current of nitrogen is passed through the furnace during the test to minimize oxidation of the coal and to sweep out the volatile products, preventing their decomposition on the upper, cooler parts of the setup. The balance reading is taken each minute during the test and thus a complete record is obtained of the weight of the volatile products evolved per minute over the temperature range studied. When the temperature of 550°C. is reached the sample is cooled in the furnace and weighed as a check for total weight loss during the test.

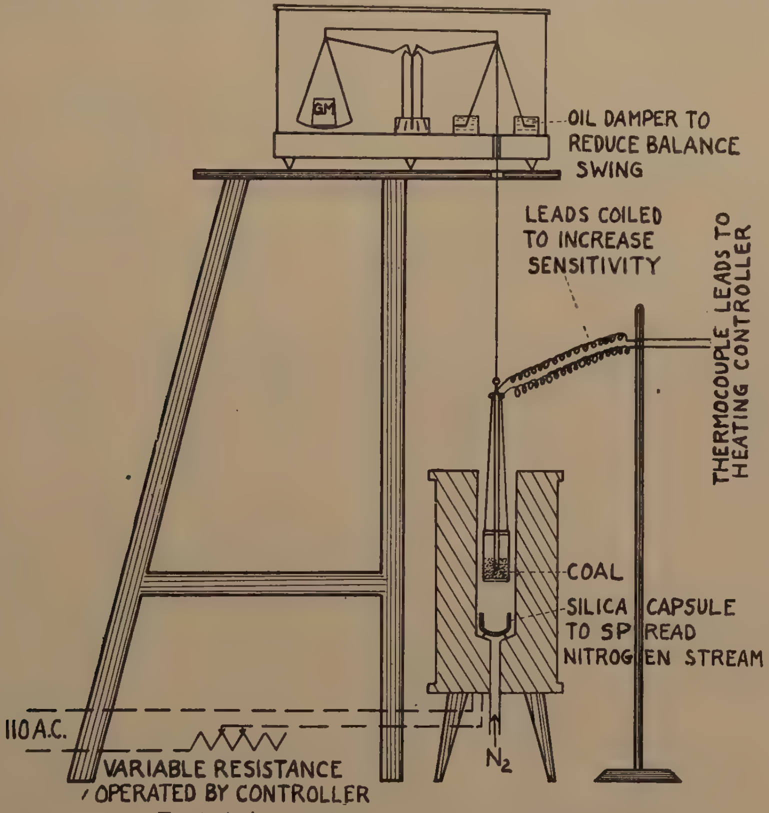
GM
OIL DAMPER TO REDUCE BALANCE SWING
LEADS COILED TO INCREASE SENSITIVITY
THERMOCOUPLE LEADS TO HEATING CONTROLLER
COAL
SILICA CAPSULE TO SPREAD NITROGEN STREAM
110 A.C.
VARIABLE RESISTANCE OPERATED BY CONTROLLER
N_2

FIG. 3.—APPARATUS FOR COAL VOLATILIZATION.

The data are reported as shown in Figs. 7–12. The rate of loss of weight in milligrams per minute per 5 grams of coal is plotted against temperature. The rate of volatilization increases slowly from 330° to 420° or 430° and then increases rapidly up to a maximum rate at 460° to 480°. Then the rate drops off somewhat but continues to maintain an appreciable value to the upper temperature limit of the test. The main difference in the rate of evolution of volatile matter for the high-volatile and the low-volatile coals appears during the plastic temperature range. Up to 400° the curves for the two coals are much the same and above 500° the rate of evolution of volatile matter is of the same order, but between 400° and 500° the high-volatile coal shows a much greater evolution of volatile products and the maximum rate is much greater. For most blends of coking coals of 26 to 31 per cent volatile matter, the volatilization curves are much the same, and although the temperatures of significant points may vary, the maximum rate of evolution of volatile matter has been found to occur during the plastic temperature range. The maximum for some straight low-volatile coals occurs after the plastic temperature range is passed. The several lesser maxima and minima of the volatilization curves have not been examined thoroughly. Duplicate tests have shown that they are reproducible for individual coals. It appears that they may reflect the evolution of various species of organic compounds or their decomposition points much the same as do the intervals of constant temperature on distillation curves of mixtures.

Measurement of Shrinkage of Charge during Carbonization

A change in volume of the coke occurs through shrinkage of the coke between its formation temperature, the end of the plastic range, and the maximum temperature attained by it. The shrinkage of the coke as it is heated to high temperatures has been studied by the the U. S. Bureau of Mines.[51] By several test methods those workers arrive at the same conclusion; namely, that all cokes, no matter what rank of coal is used in their formation, shrink to about the same extent, 10 per cent on a linear basis, on heating from 500° to 1000°C. This conclusion has been criticised in the literature,[52] and it does seem unreasonable both from theoretical principles and from operating experience that all cokes should shrink to the same extent between formation temperature and 1000°C.

To examine this conclusion, another method of test was devised for measuring coke shrinkage on heating the coke from its formation temperature to 1000°C. Several coals were carbonized in the movable-wall test oven until only an inch or so of uncarbonized coal remained in the center of the charge, between the plastic seams. Thus, when the coke was pushed, full-length pieces of coke were known to vary in temperature attained during the coking period from high-temperature coke to coke just emerging from the plastic state. By means of thermocouples inserted in the charge at 1-in. intervals across the oven, the temperatures attained by the coke at various distances from the oven walls were known. Full-length coke pieces were cut into monoliths approximately one inch square and then cut into sections varying in thickness from about 3/16 in. at the plastic seam end to 1 in. at the cauliflower end. The pieces were then tested in the apparatus shown in Fig. 4. Up to 500°C. the rate of heating was about 10° per min. and above 500° the rate of heating was approximately 5° per min. The change in the thickness of the coke piece tested was shown on the dial. The lever system used allowed a variation of sensitivity by change of the lever ratio. Thus, the dial, each division of which equals 1/1000 in. at a 1:1 ratio of the lever arms would indicate a change in length of 1/10,000 in. when the arm ratio was set at 10:1. To ensure satisfactory operation of the instrument, tests were made of the apparatus when it contained no sample and also tests were made of the linear thermal expansion of steel.

Fig. 5 shows the results of tests of the shrinkage of coke samples from several coals. These curves are for samples of coke taken at the plastic seam end of the coke pieces and each sample was approximately $\frac{3}{16}$ in. thick. Two points are immediately evident: (1) shrinkage does not begin at the same temperature for each sample, and (2) the total shrinkage on heating up to 1000°C. varies from coal to coal, being less for coke made from the low-volatile coal. The curve for the sample made from coal of 33 per cent volatile matter content appears out of line and this may be due to the selection of a sample of slightly higher temperature coke. The difficulties of the test are shown by the fact that the temperature gradient in the coke adjoining the plastic seam is approximately 300° for the first inch. Thus, even a sample $\frac{1}{8}$ in. thick represents 30° to 40° difference in temperature at the two faces. These tests show that there is a distinct measurable

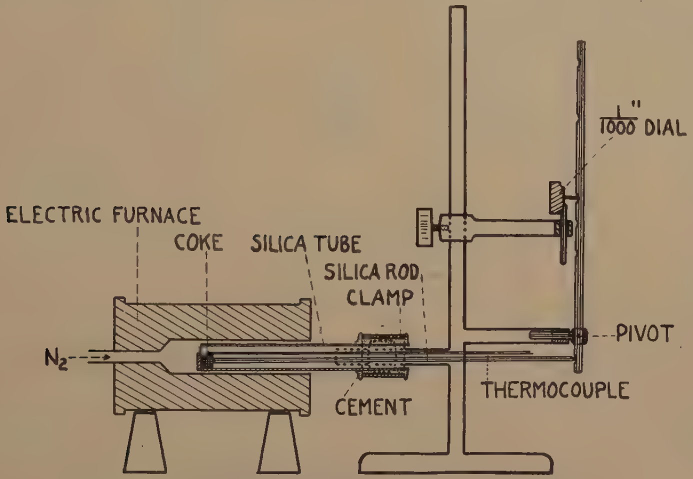

FIG. 4.—APPARATUS FOR TESTING COKE SHRINKAGE.

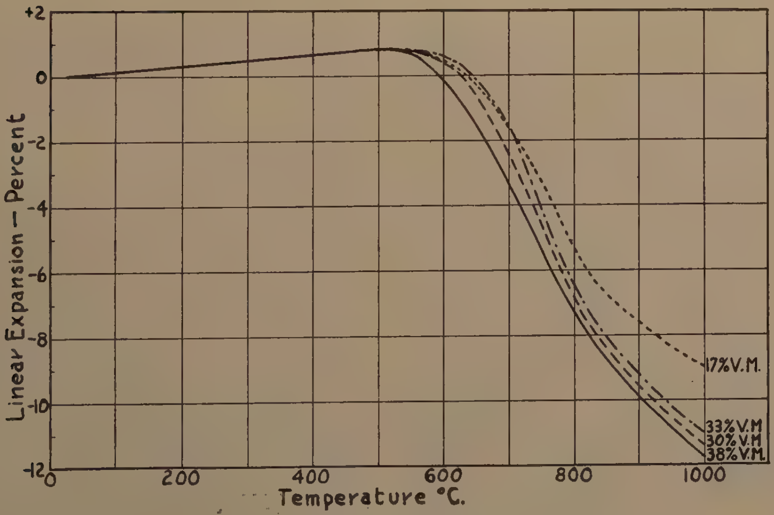

FIG. 5.—SHRINKAGE OF COKES MADE FROM DIFFERENT COALS.

difference in the shrinkage of cokes made from different coals. The coke from the 17 per cent volatile coal gave a total shrinkage of 8.95 per cent while that from the 38 per cent volatile coal showed a shrinkage of 12.34 per cent. The difference amounts to 38 per cent but this is not in direct proportion to the original volatile matter contents of the two coals or in proportion to the volatile matter left in the semicoke at the end of the plastic range. For 17¼ in. of coke, the coke from the low-volatile coal would shrink 1½ in. while that from the high-volatile coal would shrink 2$\frac{3}{16}$ in., a difference of $\frac{11}{16}$ inch.

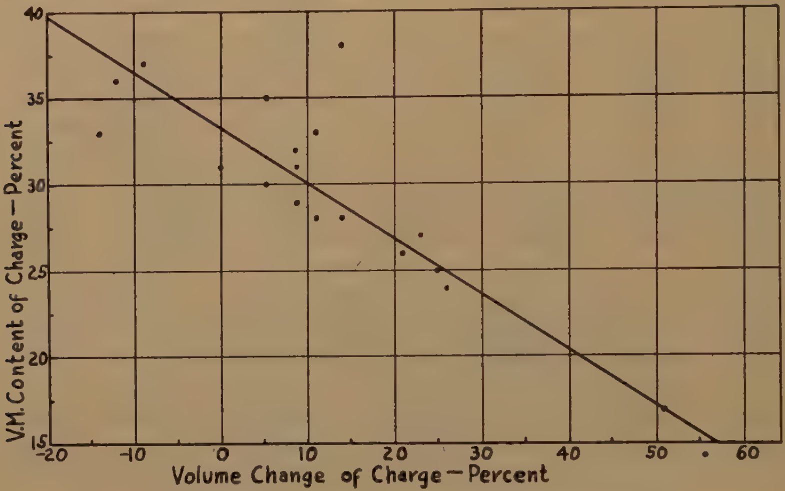

FIG. 6.—VOLUME CHANGE OF CHARGE DURING CARBONIZATION.

A change in volume occurs also over the plastic range. Coal at one bulk density at the beginning of the plastic range undergoes a weight loss during the plastic interval and emerges as semicoke at another bulk density. It is not possible at present to measure or to calculate exactly this volume change because the bulk density of the coal as it enters the plastic stage is not known. The bulk density may be quite different from the average figure for the bulk density of the coal in the oven. However, it is possible to calculate the over-all change in volume that takes place during carbonization. Since the volume change due to coke shrinkage can be measured, the change in volume of the charge due to changes up to the end of the plastic range can be calculated. This calculation requires data on the specific gravity and the bulk density of the coal, the volatile matter weight loss during carbonization or the coke yield, and the specific gravity of the coke. The calculation of the volume change for various amounts of volatile matter in coal blends is given in Table 1 and shown graphically in Fig. 6.

The calculations in Table 1 show the volume change for the entire carbonization period, not just the change occurring during the plastic interval. Since it has been shown that the volume change due to coke shrinkage is much the same for ordinary blends of coking coals, the major differences in the volume change of the charge must occur before the solidification of the plastic coal. Changes in the volume of the uncarbonized coal—i.e., compression of the coal due to pressure exerted on the coal by reactions in the plastic seam, or slumping of the charge due to insufficient pressure to support the coal—appear to be small.

For blends of coking coals ordinarily used in coke ovens, the amount of sponge formed is small, and this indicates that there is little slumping of the charge. Experimental work indicates that some pressure must be present during the progress of the plastic seams toward the oven center to obtain good coke, and this pressure minimizes slumping. Thus, change in the bulk density of the charge due to slumping of the coal appears to be small. It is doubtful whether the compressibility of pulverized coal as charged into coke ovens would vary appreciably for different blends of coal. For ordinary blends of coals the pressure exerted upon the coal between the plastic layers is small—it is estimated to be 1 lb. per sq. in. or less over the greater part of the coking period. Therefore it seems reasonable that the average bulk density of the coal does not change appreciably before the coal becomes plastic. The major part of the difference in the volume change in the charge during carbonization, for different blends of coal, seems to be due to changes that occur during the plastic stage.

As nearly as possible the data of Table 1 are taken from commercial coke-oven operations. The specific gravity and the bulk density of the coal are assumed identical for all the coals, for simplicity. Lack of accurate information of the true specific gravity of blends of coking coals makes this assumption necessary. The apparent specific gravity of the coke is that of full-length pieces from 17¼-in. Becker ovens operated on approximately 16-hr. coking schedules, except for the first two coals for which the data are taken from results in the movable-wall test oven and corrected as nearly as possible to agree with results for commercial ovens. The coke yield is an estimated value obtained by laboratory determination. It is believed that the apparent specific gravity of the coke and the true specific gravity of the coal should be used because the granular coal as charged is compressible while the porous coke is not compressible under the pressure conditions in modern by-product coke ovens.

From the table, it is immediately evident that if the specific gravity of the coal is a constant and the bulk density is con-

TABLE 1.—*Calculation of Volume Change of Coal during Carbonization*

Volatile Matter of Coal as Charged[a]	Sp. Gr. of Coal (Assumed)	Charge, Bulk Density, per Cu. Ft.	Cu. Ft. Solid Coal per Cu. Ft. of Charge	Sp. Gr. of Coke	Coke, Bulk Density per Cu. Ft.	Coke Yield, Per Cent	Weight of Coke per Cu. Ft. of Coal	Volume of Coke per Cu. Ft. of Coal	Volume Change per Cu. Ft. of Charge	Volume Change, Per Cent
17	1.40	50	0.57	0.80	50	85	43	0.86	+0.29	+51
24-B	1.40	50	0.57	0.89	56	79	40	0.72	+0.15	+26
25-B	1.40	50	0.57	0.90	56	79	40	0.71	+0.14	+25
26-B	1.40	50	0.57	0.91	57	78	39	0.69	+0.12	+21
27-B	1.40	50	0.57	0.89	56	77	39	0.70	+0.13	+23
28-B	1.40	50	0.57	0.93	58	76	38	0.65	+0.08	+14
28-B	1.40	50	0.57	0.97	61	76	38	0.63	+0.06	+11
29	1.40	50	0.57	0.98	61	75	38	0.62	+0.05	+ 8.8
30	1.40	50	0.57	1.01	63	74	37	0.60	+0.03	+ 5.3
31-B	1.40	50	0.57	0.96	60	74	37	0.62	+0.05	+ 8.8
31	1.40	50	0.57	1.04	65	74	37	0.57	0	0
32	1.40	50	0.57	0.95	59	73	37	0.62	+0.05	+ 8.8
33-B	1.40	50	0.57	0.91	57	72	36	0.63	+0.06	+11
33	1.40	50	0.57	1.14	71	72	36	0.49	−0.08	−14
35	1.40	50	0.57	0.93	58	70	35	0.60	+0.03	− 5.3
36	1.40	50	0.57	1.11	69	69	35	0.50	−0.07	−12
37	1.40	50	0.57	1.04	65	68	34	0.52	−0.05	− 8.8
38	1.40	50	0.57	0.84	53	67	34	0.65	+0.08	+14

[a] B indicates blend of coals.

stant, the volume change during coking is dependent upon the coke yield and the specific gravity of the coke. As the volatile matter content of the coal charged increases, the coke yield decreases. No general relation is evident for the trend of the specific gravity of the coke with change in the volatile matter content of the coal charged. If the differences in coke shrinkage are included in the calculation, the low-volatile coal, showing a large increase in volume, would show even greater difference in volume change than the rest of the coals.

Some interesting observations may be made from these results. The low-volatile coal increases in volume from 0.57 to 0.86 cu. ft. of solid material per unit volume of oven space, or a volume increase of over 50 per cent. The remaining volume, 0.14 cu. ft. per unit volume, must suffice for the free space in the oven, such as shrinkage space between coke pieces, the center dividing line, open pores, etc., and does not seem sufficient. Auvil and co-workers (ref. 50, Figs. 7–12 and Table 5) have shown that the space taken up by superficial shrinkage—i.e., shrinkage space between coke pieces—was 19.9 per cent, or 0.20 sq. ft. per square foot of oven-wall area. Corresponding volume of oven space, omitting coke shrinkage normal to the oven wall, is 0.20 cu. ft. per unit volume of oven space. Thus, it is not surprising that low-volatile coals cause serious damage to coke-oven walls. While the calculated final volume is 0.86 cu. ft. of solid coke per unit volume of oven space, the volume is even greater at the time of the disappearance of the last plastic coal. Shrinkage of the coke in the oven center causes a decrease in volume to 0.86 cu. ft. per unit volume of oven space.

Similarly, the 33 per cent volatile coal shows only 0.49 cu. ft. occupied by coke per unit volume of oven space, or the oven is only half full of coke. This apparent anomaly is explained when the free space in the oven is totaled. Shrinkage space between coke pieces, open pores, the center dividing line space, the deep V at the top of the charge, slumping of the charge and sponge formation during coking, all contribute to the free space in the oven.

It may be noted that the change in the height of the charge during coking is not included in these calculations. The effect of slumping is more noticeable for the higher volatile blends. Thus, the 38 per cent volatile coal, showing a calculated volume increase, actually would show a volume decrease if the change in the height of the charge were included in the calculation.

These calculations also emphasize the importance of the plastic state in coke formation. The relationships within the plastic state determine to a large extent the volume change of the charge. Although other factors are of influence, generally when the fluidity of the coal blend remains low the apparent specific gravity of the coke is low. Blends showing high fluidity give well-fused cokes with high apparent specific gravity. Since most coal blends going to coke ovens fall within a relatively narrow range of volatile matter content, perhaps 26 to 31 per cent, the variation in coke yields is not extreme and the variation in coke gravity, primarily a function of the plastic properties, is a larger determining factor in the volume change of the charge during carbonization.

Explanation of Types of Pressure Curves Obtained with Movable-wall Test Oven from Laboratory Test Data

The large movable-wall test oven has been used for approximately 300 tests, and for many of these tests Gieseler plastometer determinations were made on samples of the coal charged. A considerable number of volatilization tests were made on the same coals. Proximate analyses were also available. An explanation has been derived from these data to account for the different pressure curves obtained.

TABLE 2.—*Criteria for Estimating Degree of Pressure Development*

Type of Pressure Curve	Initial Softening Temperature, Deg. C.	Temperature of 50 Div. per Min. on Ascending Portion of Curve, Deg. C.	Temperature Interval between Initial Softening and 50 Div. per Min.	Volatilization Rate at 50 Div. per Min.	Maximum Fluidity, Div. per Min.	Temperature of 50 Div. per Min. on Descending Portion of Curve, Deg. C.	Solidification Temperature, Deg. C.	Temperature Interval between 50 Div. per Min. and Solidification	Avg. Pressure between 2nd and 6th Hours of Coking, Lb. per Sq. In.	Peak Pressure when Plastic Seams Meet, Lb. per Sq. In.
I	352	418	66	8.5	15,000	477	490	13	0.3	0.3
II	362	428	66	11	2,500	476	495	19	0.9	1.2
III	371	427	56	9.0	600	473	488	15	1.1	0.8
IV	354	433	79	12	7,000	478	502	24	1.0	2.8
V	378	450	72	16	150	477	504	27	1.8	4.0
VI	428	[a]	[a]	[a]	45	[a]	506	78		5.9

[a] This straight low-volatile coal did not attain a fluidity at any time of 50 divisions per minute; consequently the entire plastic range is a region of high resistance to the escape of volatile matter.

Since all volatile matter produced during the plastic stage must pass out of the plastic seam, the degree of plasticity existing for various rates of volatilization will determine the amount of pressure developed. The resistance to escape of volatile matter decreases as the fluidity increases. Whether this resistance is a direct function of the fluidity of the coal system or is an indirect function of the fluidity is not known. The resistance may be more directly related to some property of the fused coal other than the rheological behavior, such as the interfacial tension between the fused coal, the gases, and the inerts. From the laboratory tests made so far in this work, it appears that the resistance to escape of volatile matter out of the plastic seam can be related directly to the consistency of the plastic system. The pressure, then, is a function of the fluidity and of the rate of volatile matter evolution. At each point across the plastic layer the amount of volatile matter passing the point, in relation to the fluidity of the system at that point, causes an increment of pressure. The total pressure is the summation of the increments of pressure for all points across the layer.

Calculation of the pressure in this manner is a difficult process. Mathematical computation of the pressure does not appear possible at present because the limits for the summation cannot be defined. A simple expedient has been found whereby this calculation is unnecessary. The greatest resistance to escape of volatile matter occurs where the fluidity is lowest; i.e., at the sides of the plastic seam where the coal is just softening or the plastic coal is solidifying to semicoke. It has been found that if the temperature in the plastic interval at which the fluidity of the system is 50 divisions per minute (1.5×10^5 poises) is taken as the limiting value of the low fluidity range, the extent of the temperature interval between this value and the solidification is a measure of the resistance of the plastic layer to the escape of the major portion of the volatile matter out of the plastic seam. On the coal side of the plastic seam, the temperature at which the plastic coal attains a value of 50 divisions per minute and the relationship of the rate of volatilization at that temperature is a measure of the resistance of the coal side of the plastic layer to escape of volatile matter.

For most coal blends the maximum fluidity is great enough, in comparison with the criterion of 50 divisions per minute, that the resistance of the more fluid part of the plastic layer may be assumed negligible compared with the resistance during the temperature interval of low fluidity on each side. If the maximum fluidity of the system never becomes great, the fluidity of the whole plastic interval must be considered. It has been found that if the maximum fluidity exceeds about 500 divisions per minute (1.7×10^4 poises) the over-all fluidity in the layer need not be considered and only the temperature interval of low fluidity at the sides of the plastic layer is required to explain pressure development.

The regions in the plastic layer where the fluidity is less than 50 divisions per minute may be termed the primary sources of carbonization pressure. There are several contributing factors, such as the charge shrinkage as shown previously, bulk density of the charge, presence of inerts, etc., but these are known to the coke-oven operator and the knowledge of the effects of these factors can be employed to modify any observations of the magnitude of the pressure due to the primary sources of pressure. A summary of the criteria for each of the examples used for the six classes of pressure curves is given in Table 2.

Fig. 7 shows the plasticity-volatilization relationships for type I pressure curves. There is a wide temperature region of low fluidity between the beginning of plasticity and the temperature at which the fluidity of the system becomes greater than 50 divisions per minute, 418°C., but the rate of volatilization during this period is very low. Before the rate of volatilization becomes rapid, the fluidity increases greatly. Thus, there is little cause for pressure. On the coke side of the plastic layer, the temperature interval of low fluidity, 50 divisions per minute to the solidification point, is only 13°, so that although a large amount of volatile matter is passing through this region there is little resistance to its escape and there is little cause for pressure development. It is interesting to note that these relationships may be used as evidence for the direction of flow of gases out of the coke oven. Flow toward the interior of the oven would need to pass through approximately a 65° interval of low fluidity while the flow toward the coke side passes through only a 13° interval. This indicates that, at least for coals of this type, the major portion of the volatile matter must pass out of the plastic seam to the coke-oven wall and not toward the center of the oven. Further, at the end of the plastic state, approximately one half of the original volatile matter of the coal is still present in the coke and is evolved principally as permanent gases as the coke is heated to become high-temperature coke. This volatile matter certainly would not pass through the entire plastic seam to pass into the center of the oven. Further reason for lack of pressure development is the large shrinkage of the charge during carbonization. The high fluidity of the coal is a factor in making a well-fused coke of apparent specific gravity often over 1.0, and the coke yield from such coal is lower than for usual blends of coking coals. Thus, the change in the volume of the charge also would tend to reduce the pressure development.

The coke obtained in the commercial carbonization of coals giving type I pressure curves is well fused, with high apparent specific gravity, but tends toward fingery quality and often a considerable amount of spongy coke is formed.

Pressure curves of type II have viscosity-volatilization relationships shown in Fig. 8. The over-all fluidity is somewhat less than in the previous case. The temperature region of low fluidity after the maximum fluidity is passed is 50 per cent greater. The temperature region of low fluidity before the maximum is attained extends to higher temperatures, being 428°C. in the example used, and the volatilization in this interval becomes greater than for the previous case. All these factors point to a moderate pressure formation of 1 lb. per sq. in. or so at normal bulk

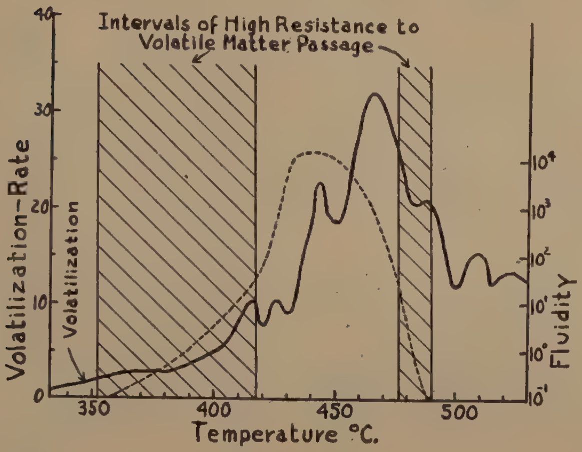

FIG. 7.

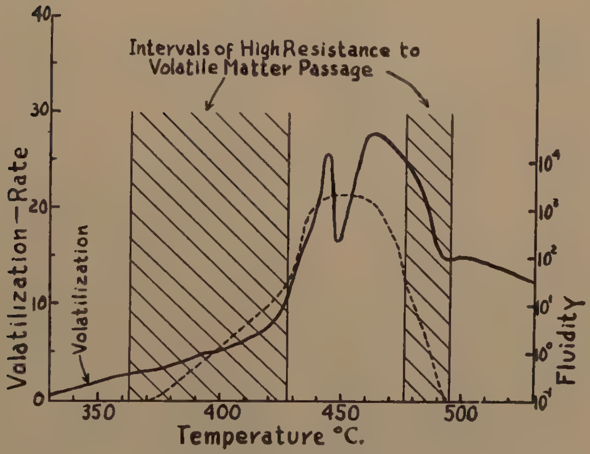

FIG. 8.

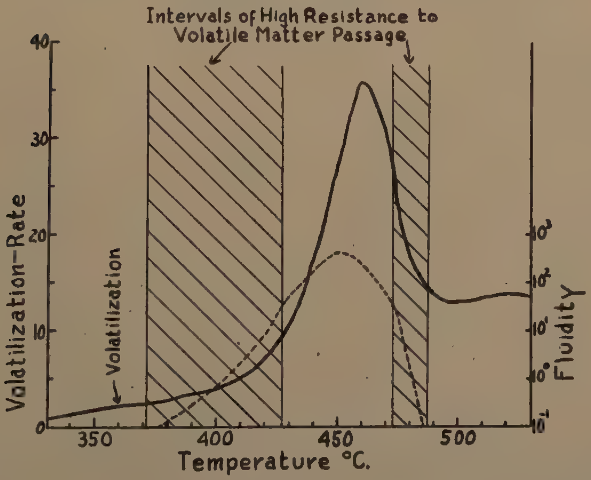

FIG. 9.

FIGS. 7–9.—FLUIDITY-VOLATILIZATION RELATIONSHIPS, PRESSURE CURVES, MOVABLE-WALL TEST OVEN.

Fig. 7, type I curves; Fig. 8, type II; Fig. 9, type III.

densities. The small peak pressure is due to the moderate rate of volatilization before the fluidity becomes more than 1.5×10^6 poises. This causes somewhat more volatile matter to pass toward the oven center and be trapped there when the plastic seams meet, and thus causes the small peak pressure.

Coke from coals giving this type of pressure curve is well fused, generally of a specific gravity of 0.9 to 0.95, and suitable for blast-furnace, water-gas, and domestic purposes. Sufficient pressure is present to minimize the tendency toward fingery coke and the formation of spongy coke. The volatile matter content, 26 to 31 per cent, is high enough so that any expansion of the charge during carbonization is compensated for the most part by the voids in the coal and by coke shrinkage, as shown by the slightly declining pressure during the intermediate stage of the carbonization period.

The pressure curves of type III have quite different viscosity-volatilization relationships because of the presence in the coal blend of large amounts of lower-rank high-volatile coal. The over-all fluidity is considerably less and this is reflected in the rapid rise in pressure during the first hour of coking,—more than 1 lb. per sq.in. (Fig. 9). After the first hour the plastic seam is relatively stabilized in thickness and the temperature gradient per unit distance across the layer is much less. Usually for coal blends giving this type of carbonization pressure curve, the solidification temperature of the coal is considerably lower than for usual blends of coking coals. In the intermediate stages of carbonization the pressure declines from the previous high value, since the rate of heating during the plastic period is much slower than during the first hour and, aside from the over-all low fluidity, the sources of pressure are small. The fluidity attains the value of 1.5×10^6 poises at 427°C. in the example given, and this occurs before the rate of evolution of volatile matter becomes rapid. On the coke side of the plastic layer, the temperature interval of low fluidity and high resistance to the escape of volatile products is small, being 15°. Coal blends of this type have relatively high volatile matter content, 30 to 33 per cent, and although the specific gravity of the coke is less than for coke from most blends of coking coals, coke shrinkage more than compensates for any expansion of the charge.

Coke from coal blends of this type is not as well fused as in the previous cases discussed. This is reflected in the lower apparent specific gravity of the coke and sometimes the appearance of pebbly seams. The coke tends toward small size with an appreciable amount of cross fracture. However, when the charge contains 35 to 50 per cent of good low-volatile coking coal, the coke is suitable for blast-furnace and domestic use.

Coal blends giving the three types of pressure curves discussed in the foregoing pages are all safe for use in commercial ovens at normal bulk densities. If the bulk density of the charge is abnormally high, the pressures are enhanced and easily may exceed the limits of safe operation for commercial ovens. From the results of many tests, it has been determined that blends of coking coals that give pressures exceeding 2 lb. per sq. in. during carbonization in the movable-wall test oven are dangerous for continued use in commercial coke ovens. The following three types of curves (types IV–VI) are for blends of coals that show pressures during carbonization exceeding this limit of safety at normal bulk densities.

Pressure curves of type IV are much like those of type II except that a high pressure occurs when the plastic seams meet in the oven center. The cause of the high pressure is shown in the viscosity-volatilization relationships given in Fig. 10. The volatilization rate becomes appreciable before the fluidity of the coal

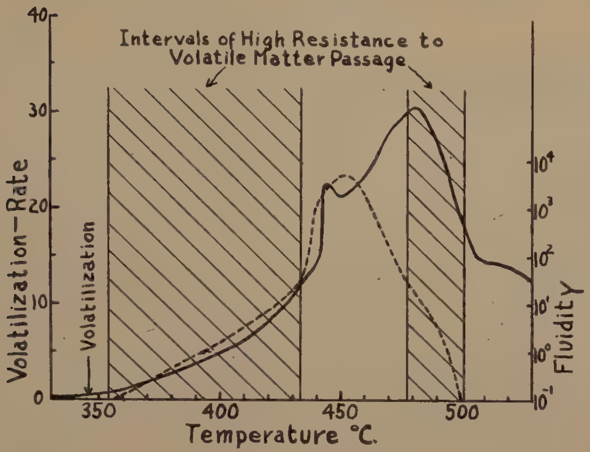

FIG. 10.

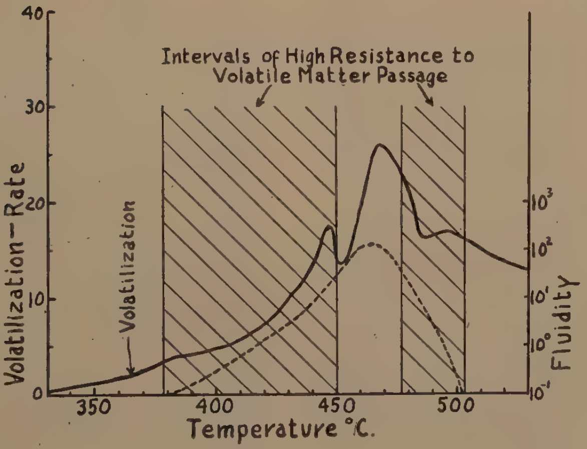

FIG. 11.

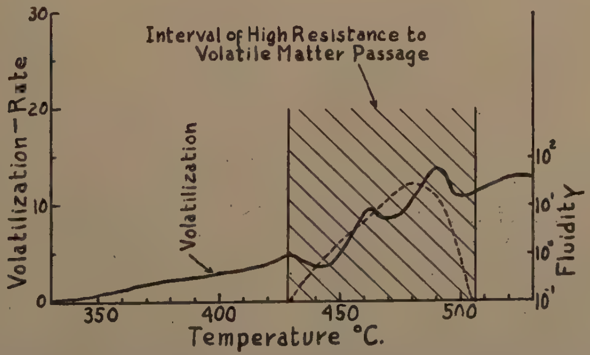

FIG. 12.

FIGS. 10–12.—FLUIDITY-VOLATILIZATION RELATIONSHIPS, PRESSURE CURVES, MOVABLE-WALL TEST OVEN.

Fig. 10, type IV curves; Fig. 11, type V; Fig. 12, type VI.

becomes greater than 1.5 × 10^5 poises, at 433°C. This tends to cause an increase in the amount of the volatile products passing into the center of the oven to be trapped there when the plastic seams meet. The temperature interval of high resistance on the coke side of the plastic seam is again small, being 24° in the example shown. This interval, in addition to the higher rate of volatilization on the coal side of the layer before the coal becomes very fluid, is sufficient to cause the pressure of approximately 1 lb. per sq. in., which exists over the principal part of the coking period, but is insufficient for the development of dangerous pressures until the plastic seams meet. The volatile matter content of this type of coal blend is again 26 to 31 per cent, as for type II curves, and the over-all fluidity of the coal during the plastic stage is sufficient for the formation of a well-fused coke of specific gravity over 0.9, so that the change in the volume of the charge during carbonization is small, compensated by the coke shrinkage, as shown by the small decline in pressure during the intermediate part of the coking period.

The physical characteristics of the coke are much the same as for coal blends giving type II pressure curves and the coke is satisfactory for most purposes. This type of coal blend is perfectly safe for use in ovens until the plastic seams meet. When the seams meet, there is a short interval of high pressures of dangerous magnitude. At abnormally high bulk densities the pressures are enhanced and are liable to cause damage to coke ovens throughout the carbonization period. This type of curve is an example of a coal blend that by ordinary methods of assessment would probably be judged satisfactory for use in coke ovens. Only a test in the movable-wall oven or the employment of viscosity-volatilization relationships as shown in this paper would show its dangerous characteristics. Coal blends giving this type of pressure development in the movable-wall test oven are most often found to be blends that have been used for some time commercially, which occasionally cause "stickers."

Type V pressure curves are for coal blends not commonly used in commercial ovens. The particular test used for illustration is for a blend containing 35 per cent low-volatile coal, and high pressures often result when the amount of low-volatile coal is as high as this. The viscosity-volatilization relationships for this type of curve, Fig. 11, explain why the high pressures develop. The over-all fluidity of the blend is much lower than usual. The fluidity never becomes much more than 10^5 poises, so that the entire plastic range offers considerable resistance to the escape of volatile products. In the example given the temperature interval of low fluidity on the coke side of the plastic layer, 27°, is not in itself a cause of dangerous pressures. However, on the coal side of the plastic seam, the coal does not exceed 1.5 × 10^5 poises until a temperature of 450°C. is reached, and at that temperature the rate of volatilization is rapid. Thus, the coal side of the plastic layer causes pressure during the entire coking period. When the plastic seams meet, the very high temperature that must be reached before the coal exceeds 1.5 × 10^5 poises in fluidity, and which has caused much more than the usual amount of volatile products to pass into the oven center, signifies that high peak pressures will develop. The example given shows a jagged peak of about 4 lb. per sq. in., and this pressure continues for an appreciable length of time. Possibly in this example the heating was poor and all parts of the plastic seam did not meet at the same time. Usually, the peak pressure shown when the plastic seams meet last for less than 30 min., while in this example it extends nearly one hour.

There is another case of this same type of pressure curve in which the pressure is due to other causes. When the plasticity test shows a wide region of low fluidity after the maximum fluidity is passed and before the plastic coal solidifies to semicoke, the pressure during the entire coking period is high. The maximum fluidity may be high and the relationships on the coal side of the plastic layer for viscosity and volatilization may be such as to cause only small pressure development. The wide region of low fluidity on the coke side of the plastic layer causes high pressures throughout the carbonization period until all the charge has passed the plastic stage, because the volatile matter evolved within the layer must pass through this wide temperature region of low fluidity. When the plastic seams meet, the increase in heat transfer to the plastic coal causes an increased rate of decomposition of the coal and consequently a peak pressure is shown, as for the curve for the coal blend containing large amounts of low-volatile coal. Several coals have been found which on the basis of maximum fluidity and volatile matter content would ordinarily be judged safe for use in commercial ovens, but which show high, dangerous pressures when tested in the movable-wall oven, and whose viscosity-volatilization relationships show these causes for the dangerous pressure development.

Many low-volatile coals when coked by themselves give pressure curves of type VI. Plasticity-volatilization relationships, illustrated by Fig. 12, show why such coals cause damage to by-product coke ovens. Usually the fluidity is never greater than 1.5×10^5 poises throughout the plastic period and often remains below 10^6 poises. This signifies that the entire plastic layer offers high resistance to the escape of volatile matter out of the plastic seam. Although the amount of volatile matter evolved during the plastic stage is much less than in usual blends of coking coals, the high resistance in the plastic layer to its escape creates high pressures. Further, as shown in Table 1, the charge is expanding throughout the carbonization period much more than could be compensated for by coke shrinkage. All factors point to dangerous pressures, which become increasingly higher as carbonization proceeds.

Method of Assessment of Laboratory Data for Predicting Carbonization Pressures

The methods presented in this paper have not been developed to the point where numerical data obtained in laboratory tests can be substituted in a simple formula from which a numerical result can be obtained and used to assess a blend of coking coals in respect to carbonization pressure. However, the examination of laboratory tests, as indicated below, can be employed to predict the relative safety of coking coals. It should be understood that any such predictions apply only to normal operating conditions

TABLE 3.—*Laboratory Analyses of Two Coal Blends from Which Carbonization Pressure May Be Predicted*

Laboratory Analysis	Coal Blend A	Coal Blend B
Proximate analysis, per cent:		
Volatile matter	27.6	28.0
Fixed carbon	67.3	66.6
Ash	5.1	5.4
Sulphur	0.68	0.78
Gieseler plastometer test:		
Initial softening, deg. C	368	364
50 div. per min. on ascending branch of curve, deg. C	436	426
Maximum fluidity, div. per min	2,400	510
Maximum fluidity, deg. C	451	444
50 div. per min. on descending branch of curve, deg. C.	476	467
Solidification, deg. C	496	483
Temperature interval of low fluidity on coke side of plastic seam, deg	20	16
Volatilization test:		
Rate of loss at temperature at which fluidity is 50 div. per min. on ascending branch of curve, mg per min. per 5 grams of coal	12.0	10.5
Maximum rate of loss of volatile matter, mg. per min. per 5 grams of coal	35.5	34.0
Temperature at which maximum rate occurs, deg. C	460	450
Volatile matter lost up to end of plastic range, per cent of total volatile matter of coal	59.5	60.5
Movable-wall test-oven result, maximum pressure, lb. per sq. in	3.10	1.05

of a coke oven, and would need to be modified if the coal blend is to be carbonized under unusual conditions in respect to flue temperatures, bulk density, pulverization, moisture, use of oil, presence of inerts, or other factors.

Table 3 contains laboratory data for two typical blends of coking coals, which may be obtained from less than one pound of each coal. The numerical values are taken from actual tests, except for the volatilization data; the latter are figures taken from tests on similar coals, because tests were not available for the coals used in the examples. The predictions made in the following assessments agree with tests of the coals in the movable-wall test oven.

Assessment of Coal A

The volatile matter content of 27.6 per cent indicates a coke yield of 76 per cent. The maximum fluidity attained (2400 divisions per minute) indicates a well-fused coke with an apparent specific gravity about 0.95. These data show that there is little evidence for much change in the volume of the charge during carbonization (Table 1).

On the coke side of the plastic range the temperature interval of low fluidity is 20°. On the coal side, the limiting value of 50 divisions per minute fluidity is not attained until 436°, where the rate of volatilization is 12.0 units. These results indicate that relatively high pressure would be present throughout the coking period. In addition, the limit of 436°C. on the coal side of the plastic layer for the low fluidity and the volatilization rate at this temperature indicate that sufficient volatile matter would pass into the center of the charge to cause high peak pressure of dangerous magnitude when the plastic seams meet.

The temperature of maximum fluidity, 451°C., is normal for coal blends of this type. The maximum rate of loss of volatile matter, 35.5 units, at 460°C. is also in agreement with tests on other coals. The ash content, 5.1 per cent, is not high enough to relieve the pressure to a great extent. Interpretation of the laboratory data as shown indicates that coal blend A is dangerous to use in modern coke ovens under normal operating conditions. Tests in the movable wall oven showed a maximum pressure of 3.10 lb. per sq. in. when the plastic seams met near the end of the test.

Assessment of Coal B

A volatile matter content of 28 per cent, indicating a coke yield of 76 per cent, and a maximum fluidity of 510 divisions per minute, indicating a coke gravity of about 0.90, signify a small increase in the charge volume during carbonization, but not of dangerous magnitude. The end of the plastic range, 483°C., is relatively low, so that coke shrinkage would be greater than for coal blend A and thus would reduce the tendency for pressure development.

The temperature interval of low fluidity on the coke side of the plastic seam is only 16° and on the coal side the low fluidity ends at 426°C., where the volatilization rate is only 10.5 units. These data indicate little cause for high pressure during the carbonization period. The relatively low maximum fluidity, 510 divisions per minute, indicates that the pressure in the first 2 hr. may exceed 1 lb. per sq. in., but the low value for the end of the plastic range shows that coke shrinkage would cause a decrease in pressure in subsequent hours. The temperature of 426°C. for the limit of low fluidity on the coal side and the rate of volatile matter loss, 10.5 units, indicate that no large amount of volatile matter would pass into the center of the charge to cause high peak pressure when the plastic seams meet.

These data for coal blend B indicate that this blend is safe to use in modern coke ovens under normal operating conditions. The movable-wall test-oven curve for this coal blend showed a maximum pressure of

1.05 lb. per sq. in. at one hour after charging. The pressure declined thereafter.

Coal blend A would show a pressure curve resembling type IV movable-wall test-oven curve, while coal blend B would be an example of type III pressure development. These two examples represent an instance for which the coal showing highest maximum fluidity would be dangerous while the coal showing much lower maximum fluidity would be safe to use under normal operating conditions.

CONCLUSIONS

In the foregoing study a classification was made of the types of pressure curves obtained from tests of various coals and coal blends in the movable-wall test oven. Determinations were made also of the plastic characteristics of the coals that give different kinds of pressure development. In addition, a study was made of the rate of volatilization of these coals through the temperature range of the plastic state.

The following conclusions have been drawn from these studies:

1. Pressure curves from the movable-wall test oven show definite characteristics for different kinds of coal and coal blends.

2. Gieseler plastometer-test curves have been found to be characteristic for each class of pressure curve.

3. The curves for the rate of volatilization up to the end of the plastic temperature range have been found to be substantially the same for ordinary blends of coking coals. The rate of volatilization for low-volatile coals is noticeably different over the plastic temperature range.

4. The temperature at which the viscosity of the softened coal system becomes 50 divisions per minute (1.5×10^5 poises) on the ascending portion of the Gieseler curve has been found to be the primary indication of the degree of peak pressure when the plastic seams meet. The extent of the temperature interval from 50 divisions per minute to solidification on the descending portion of the Gieseler curve is the primary indication of the degree of pressure before the plastic seams meet. These primary sources of pressure must be assessed together, for although each is the primary factor as stated, the other is a secondary factor modifying the first for each part of the pressure curve.

5. The findings indicate that the cause of pressure lies in the resistance to the escape of volatile matter through the portion of the coal present in a highly viscous state. Except for low-volatile coals, the actual degree of maximum fluidity attained by the coal is of lesser importance in the pressure development.

6. Although this work has covered coals of considerable variety, many different coals have not been analyzed by the procedure described. It appears possible to predict, from the results of laboratory tests requiring less than a pound of coal, the pressure a given coal or coal blend will develop during carbonization. Although there have been no serious exceptions to the method of assessment for the coals examined so far, it may be that some particular coal or coals may possess peculiar properties, so that the procedure of assessment described will require some modification.

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Increasing the Percentage Production of Large-size Coke at Fast Coking Rates

By I. M. Roberts*

(St. Louis Meeting, September 1942)

The war emergency has affected every phase of industry. The gas and coke-oven companies have sought faithfully to discharge their responsibility in this critical period and have willingly modified their operations where possible to meet the ever increasing demands made upon them. Our company made commitments for blast-furnace coke for war purposes which necessitated a reduction in coking time in order to produce the amount required to meet the new demands. At the same time there were certain customers of long standing, who had been supplied with foundry and egg-sized coke produced on fairly long coking periods, who also were engaged in war work and whose patronage we desired to retain. Coke produced on short coking rates from the coal mixture we were then using would not have been satisfactory for these customers.

The average width of our ovens is 19¾ in., which is a little wider than the more modern ovens of the present day. They were originally designed to give somewhat cool tops in order to produce a rich gas. This necessitates carrying the heats a little on the warm side so that the tops will be fully carbonized.

A reduction in coking time, as is generally known, is accompanied by a reduction in the average size of the coke produced. This is due to the greater shrinkage that takes place at the higher temperatures required to carbonize the charges in a shorter time. The net result is that the coke is more cross fractured and longitudinally fractured and breaks up into small sizes on handling. The cokes produced from the better coking coals seem to be more susceptible to fracture at high temperatures than those from poorer coking coals. A coal mixture that produces a beautiful foundry-sized coke at 30 to 36 hr. may yield a product at 18 hr. that would be so badly fractured as to be unsatisfactory for foundry practice.

Coking Characteristics of Coals

Our problem was to find some economical innovation and develop a method that would enable us to reduce the coking time and still produce a coke satisfactory to our foundry trade. However, before discussing the results of our present operations, it might be well to present a few observations on the coking characteristics of coals in general. Most coke-oven operators carbonize a blended mixture of coals. It may consist of a single high-volatile coal with a single low-volatile coal, or of a combination of several high-volatile and low-volatile coals. Medium-volatile coals of the Eagle seam are used for the production of high-grade foundry coke. Medium and high-volatile coals are often blended with one or more low-volatile coals. The possibilities are almost infinite, but the combinations are limited by such factors as cost and facilities for handling multicoal mixtures, and by other practical considerations. Every coal has characteristics that make it blend better with a particular coal or coals in preference to others. This property has

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been called "compatibility" by the Bureau of Mines. The characteristics of the individual coals are reflected in the cokes produced from their mixture.

High-volatile Coals

Some high-volatile coals when carbonized unmixed with other coals produce cokes of such extremely fingery character that they are hardly suitable even for domestic purposes. They are very dense and of almost steel-like hardness, and are of low porosity. However, these same coals, when blended with the proper low-volatile coals and carbonized under ideal conditions, produce beautiful cokes eminently suitable for foundry practice.

Other high-volatile coals of almost identical proximate analyses, carbonized alone, produce cokes of soft friable character and of comparatively high porosity. When blended with low-volatile coals they produce cokes that usually are of an open character; and while they may serve as excellent domestic fuel, they are unsuitable for a competitive foundry market. Their porosities would not vary as greatly with the changes in the proportions of high-volatile and low-volatile as would those previously mentioned containing the other type of high-volatile coal.

Low-volatile Coals

The low-volatile coals, say from 16.5 to 19.0 per cent volatile matter when carbonized alone, produce cokes that to the unaided eye appear to be very dense, yet they have a porosity of 50 to 55 per cent. While we have never been so foolish as to carbonize a charge of 100 per cent low-volatile coal in our ovens, we have made many box-coking tests on them. The boxes are of sheet metal and are nearly as wide as the oven. They have double compartments and have a capacity of 1½ cu. ft. The bottom compartment is filled with a buffer charge of coal to offset in some degree the effect of the heat from the floor of the oven. The results of these box tests, while not comparable with full-scale oven tests, are nevertheless a good indication of the coking characteristics of the coal. When blended with a high-volatile coal, the low-volatile tends to suppress the longitudinal fracture and also to open up the structure of the coke.

Besides oven temperatures and mixtures employed, other conditions have a very important bearing on the coking results, such as degree of pulverization, care exercised in mixing, moisture content. Sometimes a pronounced improvement may result in the character of a coke from greater pulverization. Coals vary quite widely in character, and our experience with our present mixture relative to the use of anthracite may not be realized with other coals.

Porosity Data from Various Blends

Table 1* gives the porosity data obtained on the cokes from blends of one low-volatile coal with three high-volatile coals and with three medium-volatile coals. The cokes were produced on full-scale oven tests.

Figs. 1–4 show some of the characteristics of cokes produced from different coals. Fig. 1 is of coke produced from a high-volatile coal similar to coal A, Table 1. The fingery character of the coke and the honeycomb sponge are very much in evidence. Sponge of this character usually indicates a good coking coal. Fig. 2 is a cut section of a piece of coke produced from 100 per cent of coal A, Table 1. This coke was very dense and of extremely fingery formation. The type of cellular structure found in the coke from the 100 per cent coal persisted to a marked degree in the coke from the mixtures of the coal with low-volatile. Fig. 3 is a cut section of the coke produced from 100 per cent of coal B, Table 1. The open character of the 100 per cent coke from this coal was also reflected in the coke from the mixtures. Fig. 4 is the coke from a 60-40 per cent mixture of coal B and low-volatile and is of an open character. Fig. 5 shows the peculiar fingery tendency of the coke from a certain medium-volatile coal. This is a picture of the 60-40 per cent coke. The cokes from the other mixtures were even more fingery. A coal of this type when blended with a low-volatile normally produces a blocky coke.

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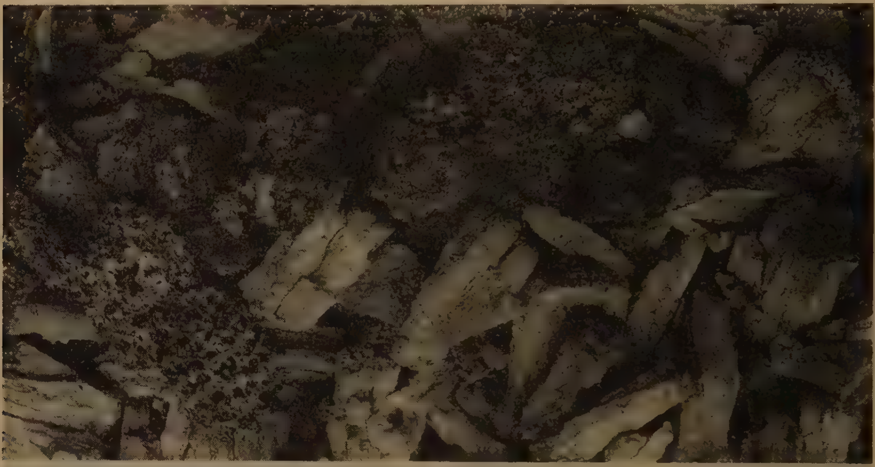
* A summary of Tables 1 to 7 is at the end of the paper.

1



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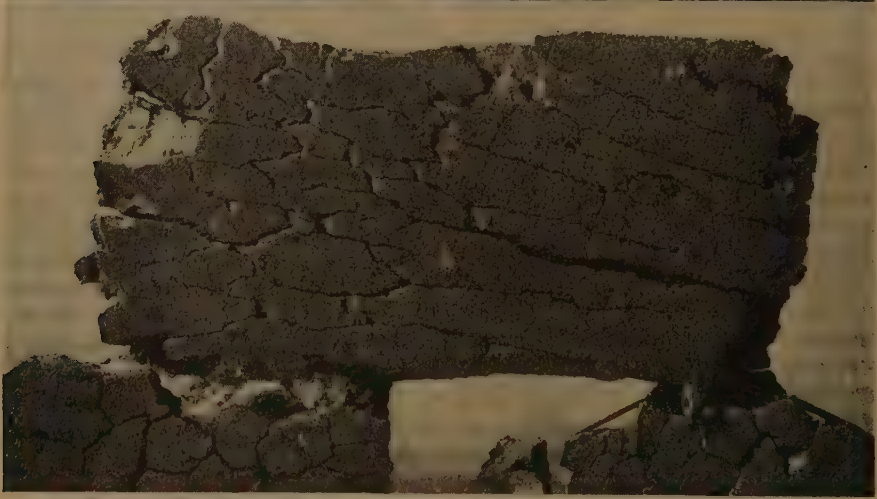
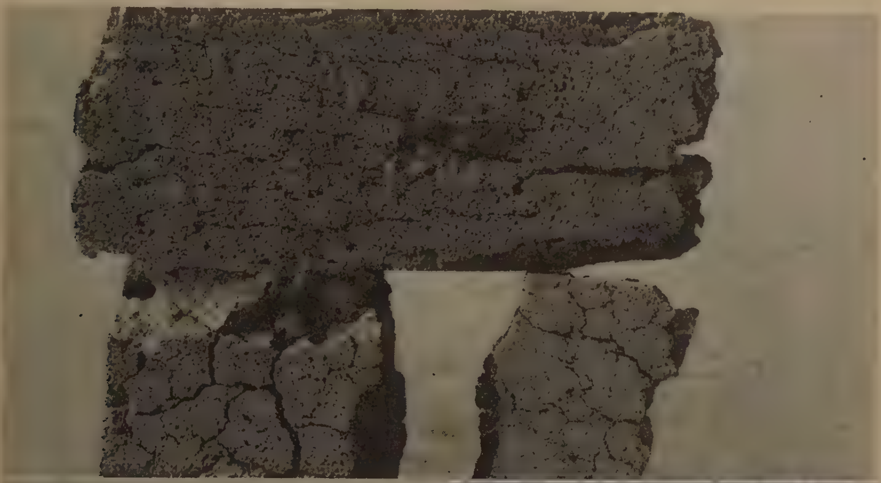


Fig. 1.—Coke from a high-volatile coal, similar to coal A, Table 1.
Fig. 2.—Coke from 100 per cent of coal A, Table 1.

Addition of Inert Material to Coal Mixture

The addition of a small percentage of inert material, such as coke breeze or other finely divided carbonaceous matter or even sand, to a coal mixture, has been found to produce a coke of a larger size than would be obtained from the mixture alone. This has been a subject of experimentation for many years, probably antedating the beginning of the twentieth century. In 1926 the Laclede company carbonized coal mixtures containing 3 and 5 per cent of pulverized coke breeze of 1⁄16-in. mesh and finer, and found that the resultant coke was of much larger size than that produced without the breeze.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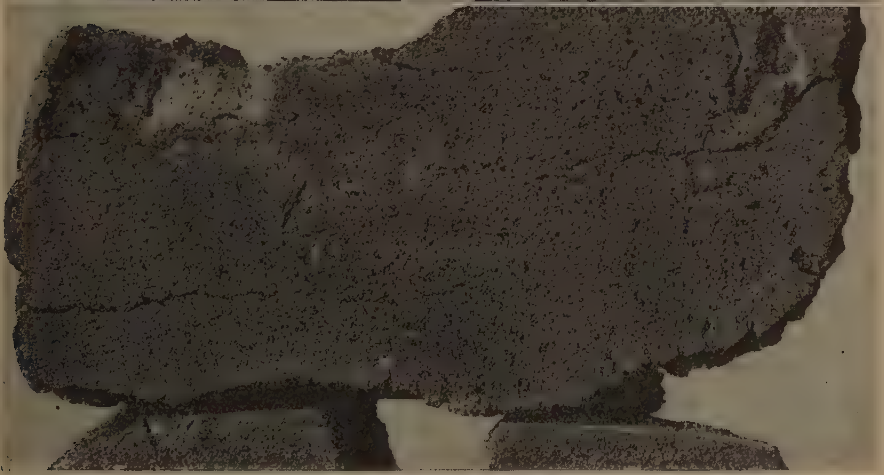
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Fig. 3.—Coke from 100 per cent of coal B, Table 1.
Fig. 4.—Coke from 60-40 mixture of coal B and low-volatile.

The *Proceedings* of the American Gas Association for 1936 record the experimental results of work of this nature done by the Philadelphia Coke Co., the Rochester Gas Co. and the Brooklyn Union Gas Co. These tests show that the size of the coke increases with the percentage of breeze between the limits of 1 to 5 per cent; also, that breeze of a size larger than minus 1⁄16 in. is detrimental to the strength of the coke.

Past experience indicated that the use of our own coke breeze properly prepared would produce the desired results. However, this would necessitate the purchase and installation of handling, drying and grinding equipment at a considerable outlay of money; if, indeed, the equipment could be obtained under present conditions. Moreover, the cost of drying and grinding coke breeze would be somewhat high, possibly $1.00 to $1.50 per ton.

Experimental Work

The problem, as previously outlined, was to find some material that could be used as purchased, or would not require any greater preparation than could be given it in the normal course of operation. Fortunately, the company has an experimental plant and a bin into which test mixtures can be run without interfering with the regular plant operation. With this equipment we were prepared to test any scheme that showed possibilities.

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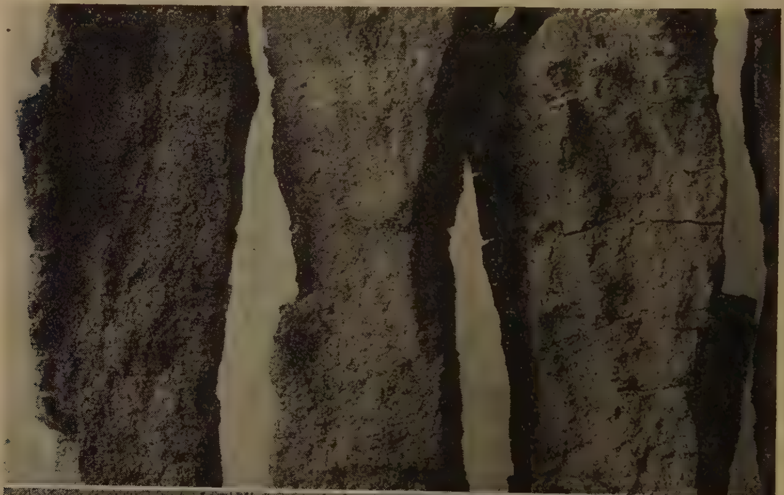

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Fig. 5.—Coke from 60-40 per cent mixture of medium-volatile coal.
Fig. 6.—Coke made from coal mixture containing Laclede coarse breeze.

The materials experimented with during a period of 4½ months were: Our regular high-volatile coal, which contains about 10 per cent of splint coal; a high-splint coal, fly-ash from boiler plants burning coke breeze as fuel, coal dust containing a high content of fusain or mineral charcoal, petroleum-coke breeze, coke breeze from various outside sources, our own breeze coarsely ground, semianthracite from the Spadra district in Oklahoma and the fine anthracite that we are now successfully using. The testing of these various materials represents a tremendous amount of work, as they were all analyzed and tested for coking results. Full-scale oven tests were made when the material could be secured in sufficient quantity and the preliminary work looked promising.

The presence of splint coal seems to give our high-volatile coal its coking characteristics; and inasmuch as a greater portion of the splint coal is found in the lump size than in the fines, it was suggested that we carbonize the lump coal for the production of foundry coke. This entailed no extra cost since in the mining operation the lump and fines are separated anyway to facilitate preparation, and are then reassembled. The use of lump coal did produce some improvement in the average size of the coke produced, but not enough to make it worth while. This was true also of the substitution of 10 per cent of an inherently high-splint coal for an equivalent amount of the high-volatile coal in the mixture.

Coal dust with a high fusain content was satisfactory with respect to the degree of fineness, but contained too much volatile matter, and so had the characteristics of a poorly coking high-volatile coal. The petroleum-coke breeze was high in both volatile matter and sulphur, and would have needed further preparation. The many coke-breeze samples from outside sources would also have needed further preparation. Only the results of materials tested on a full oven scale are dealt with in this paper. The analyses and screen tests for fly-ash from coke breeze, Laclede coarse breeze, semianthracite ⅞-in. Spadra slack, and Pennsylvania anthracite fuels are given in Table 2.*

Fly-ash

The coke produced from the charge containing 5 per cent fly-ash was promising. The yield of foundry-sized coke was 45 per cent. The surface texture of the coke was considerably rougher than that of the regular product (55 per cent high-volatile, 45 per cent low-volatile); nevertheless, it had a true coherent structure and withstood handling very well. It was rather remarkable that the coal mixture could absorb so much inert material with so little adverse effect upon the appearance and structure of the coke. The increase in ash content in the coke over that of the regular product was one objection to its use. However, there was no supply of this material available although many possible sources were investigated. The results of tests on the coke are given in Table 3.*

Laclede Coarse Coke Breeze

The coarse coke breeze as it comes from the bin was tried without much hope of success. Mixtures of the regular coals containing 4 and 6 per cent of the material were carbonized. There did not seem to be much difference between the coke produced from the mixture containing 4 per cent breeze and that containing 6 per cent except that the latter had the poor characteristics to a more pronounced degree.

The conclusion was that the coke would not have been satisfactory for the trade. While it appeared to be large, yielding 41 and 49 per cent of foundry-sized coke, respectively, nevertheless it was extremely rough and pebbly and appeared to consist of particles sintered together rather than good coherent coke. The fracture was much less extensive than that of the regular product, but the break was very irregular. The type of break showed the reason the coke held together as well as it did. The cut section of the coke (Fig. 6) shows the peculiar structure consisting of irregular areas formed by cracks somewhat after the pattern of an alligator hide.

The tumbler tests showed low stability factors for both mixtures. The shatter test was high, but has little significance considering the character of the coke. The results of the tests are given in Table 3.

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* See footnote on page 307.

Semianthracite from the Spadra District, Oklahoma

During the coal emergency shortly after World War I, the Company used in its operations some low-volatile coals from Oklahoma and Arkansas, when West Virginia coals of this type were not obtainable. They were considerably higher in ash and contained more fusain than the West Virginia coals. This partial substitution had caused a noticeable improvement in size over our regular product and it was this experience that suggested the use of a small percentage of semianthracite slack of smallest size obtainable on the market. It was hoped to obtain a coal of lower ash and sulphur content and of more constant quality than we did obtain.

The results of the preliminary tests were promising enough to try the material on a larger scale, but the later tests were unsatisfactory. The principal objections to the material were the large particle size, the high percentage of volatile matter, and high sulphur content.

The cokes produced were of slightly larger average size than those produced from the regular coals alone, but contained more slaty material, which had a tendency to cause the coke to fracture. The amount of pebbly sponge present in the charges also was objectionable. The surface texture and appearance of this coke in general were much better than that produced from the mixtures containing coarse breeze. The results of the tests are given in Table 3.

Pennsylvania Anthracite Fines

This experience with semianthracite suggested the possibility of using eastern anthracite fines, although the cost delivered was considered prohibitive. The volatile matter, ash and sulphur content were known to be lower than for the semianthracite of the Spadra district. The author knew of the use of anthracite fines in certain industries, but felt that the particle size would be too large for the purpose without grinding. However, a sample of fine anthracite, known as No. 5 buckwheat, was obtained, and as the analysis and screen test indicated possibilities, a box-coking test was made, the results of which were very encouraging. This was the Company's first experience with anthracite fines. Many samples were obtained from various sources and of different particle size, and all carefully tested. The results of the box-coking and screen tests indicated that the size of the largest particle should not be over 20 mesh and the smallest not under 100 mesh. It was not known until after the preliminary tests were completed that a large eastern coke plant had been using this material for a number of years.

Testing in the oven of two lots of No. 5 buckwheat that conformed to specifications was started April 9, 1942.

Although the moisture content of the anthracite as delivered ran about 12 per cent, it could have only slight effect on the weight per cubic foot of the coal mixture and on the porosity of the resultant coke, because the percentage of anthracite in the charge was not to exceed 6 per cent, so that the objections to the moisture were largely to paying freight on water shipped over a long distance and the handling of wet material.

A dependable source of supply was sought before starting on a program of plant tests. A visit to the anthracite region developed that there would be no difficulty in obtaining material. A brief digression may be helpful to others who may desire to use it.

In the early days of the anthracite industry there was little or no use for the small sizes below chestnut, and they were discarded as of no value. Large culm banks containing millions of tons of this rejected material* that have been accumulating over many years may be seen near the mines in the district. Rains have washed tons of this material into the streams draining the region, and some coal has been washed directly into the streams from the mining operations. This also has been accumulating for years. This small-sized anthracite is now recovered from the culm banks and streams, and also in the present mining operations, and is finding its way into various industries.

PLANT TESTS ON ANTHRACITE

Preliminary plant tests were conducted on a 21½-hr. coking schedule on mixtures of the regular coals, substituting 3, 5 and 10 per cent anthracite fines, respectively, for a proportional amount of either the high or the low-volatile coal of the mixture, as indicated in Table 4. The 21½-hr. coking rate was about the limit of speed compatible with production of satisfactory coke for the market from the regular coal mixture. As may be seen from a study of the physical characteristics of the cokes in the table, the results of the tests were very favorable. From them it appeared that the addition of 3 per cent anthracite was not quite enough; that 10 per cent was too much, and that the optimum for the operation would be 4 or 5 per cent.

The tests indicated that the production of foundry-sized coke could be approximately doubled. The coke as compared with the regular product from a two-coal mixture was appraised as follows: There was a decided absence of fracture; the surface texture was a little rougher; the surface was not quite as bright; the cellular structure was a little more uniform throughout the piece; although it had a good metallic ring when struck, its sound was not quite as sharp, indicating that the coke was a little softer; the percentage of breeze appeared to be slightly higher. The difference as a whole was not great except that the size was definitely larger. The coke was possibly a little more reactive because of its softer character, but the increase in size compensated for that. It was rather surprising that so much of the material could be added to the coal mixture and be so well absorbed that its presence could hardly be detected by one familiar with coke structure.

The fact that the coke was so free from fracture and held together so well seemed to be a result of the character of the structure. When a piece was broken the break was not square, but irregular. The surfaces formed by a break were wavy, consisting of small projections and indentations or pits. The irregularities interlocked or dovetailed together, giving the piece greater strength than that of coke produced from coals without the admixture of anthracite (Figs. 7 and 8). Those produced from mixtures of the better coking coals break squarely. The presence of anthracite certainly reduces the tendency of the charge to develop shrinkage cracks.

RESULTS ON REDUCED COKING SCHEDULES

Experience thus far indicated that the addition of 4 or 5 per cent of anthracite fines to the coal mixture would produce coke of comparatively larger average size at shorter coking times; nevertheless, it was considered advisable to conduct tests in the Piette battery on 20 and 19-hr. coking periods, respectively, so that actual data could be obtained on the coke produced. The temperatures of the ovens were increased for a 20-hr. schedule, and the first charges were placed in the ovens on May 15. The test data in Table 5 and the actual yield of foundry-sized coke show conclusively that the use of anthracite fines of the proper particle size in the coal mixture, at least within the limits of the experiments, causes a substantial increase in the average size of the coke produced, as compared with that produced from the same coals without anthracite. Similar data on the coke produced from the coal mixture without the addition of anthracite are also given in Table 5.

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* Bureau of Mines *Inf. Circ.* 7213 (June 1942) contains an interesting account of this material. In 1890 only 2 per cent of the sizes smaller than No. 1 buckwheat were marketed, whereas in 1940, 21.4 per cent found its way into the markets. The Minerals Year Book of 1940 also gives interesting data on the culm bank and dredged coal.

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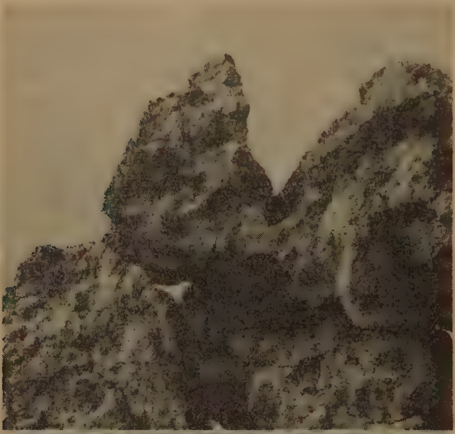

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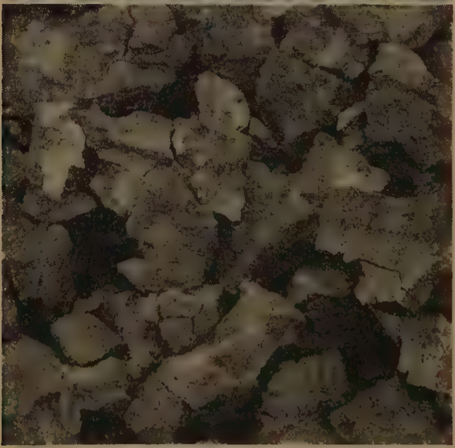

Fig. 7.—Structure of coke made from coal mixture containing small percentage anthracite.

Fig. 8.—Coke containing anthracite.

The results of the experimental work were so satisfactory that arrangements were made to operate the batteries on shorter coking schedules and charge the number of ovens required for the production of foundry coke with a mixture containing 60 per cent high-volatile coal, 36 per cent low-volatile, and 4 per cent anthracite fines. This has been done for the past two months with very satisfactory results.* The plant yield of foundry-size coke on about a 19-hr. schedule for the period was 50.5 per cent as compared with about 32 per cent for a 55-45 per cent mixture on 21½ hr. The physical characteristics of the coke from the regular production are reported in Table 6.

It is increasingly evident, as the coking time is further reduced, that a substantial foundry-size coke can still be made with the addition of anthracite fines. The success of the operation, however, apparently depends upon the selection of the proper type of anthracite fines. There seem to be a number of factors about the material itself that have an important bearing on the results and may lead to success or failure. These factors are under investigation and, it is hoped, will be the subject of a later report.

The anthracite has only about one third the gas value of the low-volatile coal and no residual value such as tar, ammonia and light oil. The total coke yield will be increased slightly and the ash content of the coke will be increased 0.3 or 0.4 per cent. Bomb tests on the regular coals and the anthracite are given for comparative purposes in Table 7.

Acknowledgment

The author wishes to express his appreciation of the kindly suggestions and help given while carrying on this work by Mr. J. D. Doherty, Research Engineer for the Koppers Coal Company.

Appendix—Summary of Tables 1 to 7†

Table 1.—Porosities of cokes produced from three (A,B,C) high-volatile coals having substantially the same proximate analysis and three (D,E,F) medium-volatile coals of the same proximate analysis. The same low-volatile coal was used throughout and all of the coals were mined in the same state. Coking time was the same for each group. The mixtures were 100 per cent high-volatile, 0 low-volatile; 80-20; 70-30; 60-40; 50-50 for A,B,C and the same of medium-volatile and low-volatile for D,E, and F. Percentage cells ranged from 29.4 to 51 for A; 52-56.4 for B; 53.2-55 for C; 42-53.3 for D; 51.4-55.8 for E; and 44.5-53.7 for F.

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* We now have nine months operation behind us (April 1943) and results are still very satisfactory.

† Tables 1 to 7 have been deposited with the

Table 2.—Proximate analyses and screen analyses of materials: (1) fly-ash from coke breeze; (2) Laclede coarse breeze; (3) semianthracite ⅞-in. Spadra slack; (4) Pennsylvania anthracite fines—used in full-scale oven tests. Proximate analysis of Pennsylvania anthracite fines: volatile matter, 5.37 per cent; fixed carbon, 80.97; ash, 13.67; sulphur, 0.65; moisture, 12.70.

Table 3.—Physical tests on cokes containing fly-ash, coarse coke breeze and semianthracite. throughout the mixtures, are in the order high-volatile: low-volatile: inert material: (*a*) 55-40-5 fly-ash; (*b*) 55-39-6 Laclede coarse breeze; (*c*) 55-41-4 L.c.b.; (*d*) 49.5-40.5-10 Spadra; (*e*) 55-40-5 S.

(1) Screen tests of run-of-oven coke from wharf belt (coking time 21½ hr.) Total percentage on +2-in. screen ranged from 81.0 for *e* mixture to 87.9 for *a*; foundry coke recovered from 32.0 per cent for *e* to 49.3 for *b*; total coke, all mixtures, 75 per cent.

(2) Screen tests of fines through 3 by 3½-in. grizzly screen. Total percentage on +1-in. ranged from 79.6 for *b* to 94.4 for *d*.

(3) Shatter tests on coke over 3 by 3½-in. grizzly screen. Total percentage on +2-in. ranged from 74.8 for *d* to 88.5 for *b*.

(4) Tumbler tests on 2 by 3-in. coke from run-of-oven samples. Cumulative percentage on +1 in. ran from 40.6 for *c* to 69.6 for *e*; +¼-in. ran from 60.7 for *b* to 77.8 for *e*. Under porosity true and apparent specific gravities are given and percentage of cells is reported as ranging from 49.4 for *e* to 51.4 for *a*.

Table 4.—Physical characteristics of coke produced during preliminary plant tests on mixtures containing anthracite fines; coking time, 21½ hr. Mixtures of h-v.-l-v. anthracite fines were made up as follows: (*a*) 52-43-5 per cent; (*b*) 53-44-3; (*c*) 58-39-3; (*d*) 57-38-5; (*e*) 47.5-47.5-5; (*f*) 48.5-48.5-3; (*g*) 40-50-10; (*h*) 50-40-10; (*i*) 50-50-0.

(1) Screen tests of run-of-oven coke from wharf belt. Total percentage of +2-in. screen ranged from 73.7 for (*i*) to 92.6 for (*h*). Foundry coke, percentage dry: (*i*) 28.0; (*c*) 35.5; (*d*) 51.8; (*g*) 58.7; (*h*) 63.8.

(2) Shatter tests on coke over 3 by 3½-in. grizzly screen. Total percentage on +2-in. ranged from 85.8 to 95.1.

(3) Tumbler on 2 by 3-in. coke from run-of-oven sample, cumulative percentage. On 1 in., 58.0 to 64.7; on ¼-in., 64.0 to 72.7 per cent. Specific gravities, true and apparent are given. Percentage cells, 51.1 to 54.3.

Table 5.—Comparative tests made in Piette battery, 20 and 19-hr schedules. For both 20 and 19-hr. the mixtures were: (*a*) h-v. 57-l-v 38-anthracite 5 and (*b*) 60-40-0.

(1) Screen tests of run-of-oven coke from wharf belt. Total percentage of +2-in., 20-hr.: (*a*) 91.1, (*b*) 79.4; foundry coke, dry basis: (*a*) 53; (*b*) 27.9. Same data for 19-hr., same order: 88.3, 78.5, and 48 and 23 for foundry coke.

(2) Shatter tests on coke over 3 by 3½-in. grizzly: 20-hr. on +2-in., (*a*) 87.3, (*b*) 84; 19-hr., on +2-in., 89.5 and 79.3.

(3) Tumbler tests (cumulative percentages), 2 by 3-in. run of oven coke, 20 and 19-hr. 20-hr., on 1-in., (*a*) 58.2, (*b*) 61.3; on ¼-in., (*a*) 68.3, (*b*) 68.8; percentage cells, (*a*) 50, (*b*) 52.4. 19-hr., equivalent data: (*a*) 60.2, (*b*) 58.1; (*a*) 67.6, (*b*) 68.8; percentage cells: (*a*) 50.5, (*b*) 52.8.

Table 6.—Physical characteristics of coke produced during regular operation on 19½ and 19-hr. coking time, also comparative data on regular mixture when on 21½-hr. coking schedule.

(1) Screen tests of run-of-oven coke from regular production sampled at head of belt to grizzly screen. Average total percentage on +2-in. screen for 19½-hr. runs of four: (*a*) 60 h-v.-36 l-v.-4 anthracite mixtures was 86.6. Equivalent figure for 19½-hr. runs of four: (*b*) 60 h-v.-40 l-v. mixtures was 70.4. Equivalent figures for 19.0-hr. runs: (*a*) 78.6; (*b*) 68.4.

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(2) Shatter tests on coke over 3 by 3½-in. grizzly. 19½-hr., (*a*) mixtures, total on +2-in., 91.2 per cent. 19-hr., (*a*) mixtures, on +2-in., 90.3; (*b*) mixture, on +2-in., 78.7 per cent.

(3) Tumbler tests (cumulative percentage). 19½-hr., (*a*) on 1 in. 58.5 per cent, on +¼-in., 67.4 per cent. 19-hr., (*b*) on 1-in. 60.5 per cent, on +¼, 68.8 per cent.

(4) Plant yields of foundry coke containing anthracite (as is) to correspond with screen tests on 19.5 coke, on various dates, (*a*) mixture: yields range from 55.1 to 57.8 per cent.

(5) Average physical test results made on coke from regular coal mixture (55 per cent high, 45 per cent low-volatile coal when on 21½-hr. coking time). Average screen test: on +2-in., 77.7 per cent. Shatter test, +2-in., 84.7. Tumbler test, +1-in. 64.6, +¼-in., 72.2.

Table 7.—Bomb tests on regular (*a*) high and (*b*) low-volatile coals, (*c*) anthracite fines, and (*d*) mixture (60 h-v.-36 l-v.-4 anthracite) containing anthracite fines. Cu. ft. gas per lb. coal carbonized, B.t.u. value of gas, B.t.u. ft. per lb. coal as gas, NH_3 per ton, tar gal. per ton, percentage coke yield are given for *a*, *b*, *c* and *d* on dry and as-received bases. Moisture is reported as received and entire above analysis is also given for anthracite fines on predried basis. Moisture for anthracite fines as received is 9.72 per cent; next highest is 3.75 for low-volatile coal.

Production and Use of Low-temperature Char as a Substitute for Low-volatile Coal in the Production of High-temperature Coke

By J. D. Price* and G. V. Woody,† Member A.I.M.E.

(New York Meeting, February 1944)

Many producers of by-product coke have spent considerable time and given considerable thought to the use of a substitute for low-volatile coal as an admixture with high-volatile coking coal for charging high-temperature by-product coke ovens. Generally speaking, there are two principal reasons for this, one being occasioned by the lack, in certain localities, of low-volatile coal delivered at a cost sufficiently low to permit its use, and another being the desirability of finding a substitute product with a delivered cost lower than the cost of the low-volatile coal. It was for the first of these reasons that the Colorado Fuel and Iron Corporation was particularly interested in this subject. This paper deals with the experimental work and the results obtained by that company in producing and using low-temperature char.

Overcoming Defects in Coke from Colorado Coals

Colorado coking coals are all of the high-volatile type and were all laid down during the Cretaceous period, which makes them something like 80 million or so years younger, geologically, than eastern coals. They behave, when coked alone, something like the Pittsburgh-seam coals. They make a very brittle, highly cross-fractured, fingery coke of fair shatter value but quite low in resistance to abrasion. Mixed with a low-volatile coking coal such as Pocahontas or certain Oklahoma coals, they make a coke that is equal to virtually any of the eastern cokes.

The inferior quality of Colorado coke has long been recognized, and a considerable amount of time, effort and money has been spent upon investigations of means for improving its physical properties. While it has been found that some improvement in quality of coke can be secured by the adjustment and control of such variables as pulverization, moisture content, oven temperature and bulk density of charge, and through the addition of certain inert materials such as pulverized coke breeze, pitch, high-volatile noncoking or semicoking coal, the benefits so gained have been comparatively small. There are no low or medium-volatile coking coals available within a reasonable freight-cost distance. Therefore a substitute for low-volatile coal was developed through low-temperature carbonization—or, more correctly speaking, through the partial devolatilization—of high-volatile coals. The blending of this lower volatile char with the high-volatile coking coal distinctly improves the quality of coke made from Colorado coal.

There is nothing especially new about this idea. In 1908 a Japanese patent was taken out by Kotaro Shimomura, whose claim was:

> The method of making a nonfingery coke out of bituminous coals, without using natural coals of low-volatile matter; according to which a certain coal is heated at a temperature between about 300°C. and about 600°C. so as to leave about 15 to 25 per cent volatile matter in the coal; this coal is mixed with the

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unheated coal in the proportion of about 20 to 35 per cent of the heated coal and 80 to 65 per cent of the unheated coal, and the mixture is heated at a temperature between 1000°C. and 1400°C.

It has been used in Germany, and Adolph Thau describes the manufacture of a semicoke for this same purpose in his book, "Die Schwelung von Braun und Steinkohle."

In this country, James G. West, in 1923, obtained a patent under which his first allowed claim is stated as:

> The method herein described of producing from high-volatile coal metallurgical coke which consists of subjecting a body of such coal to distillation and while the material under treatment is still a fused and gummy mass, cooling it, then crushing the so treated material and mixing it with another body of the same material in the raw state, and then coking the mixture to the end.

Requirements for Making Char

Almost all of the writers who have described the use of this process have agreed on one point. The char, for optimum results in improvement of the physical properties of the coke, should be made from the coking coal or mixture of coals that is later to be used in preparing the final mixture. Theoretically this reasoning is sound. It should follow, therefore, that the best coke would be produced when the char and the parent coal have both the same plastic properties, for if both have melting points within a narrow range of temperature, the fusion would be uniform and the final coke would not have the pebbly or agglomerated appearance that would conceivably result should the plastic range of the char and of the coal used in the final mixture differ. However, this point has been disproved, at least as far as Colorado coals used by the Colorado Fuel and Iron Corporation are concerned.

The first problem was to secure a suitable type of apparatus for the production of the char. Following the accepted theory that char should preferably be made from the coal that is later to be used in the blend, Shimomura's analysis of the requirements for his equipment appeared sound. These were:

1. Brisk agitation is necessary to prevent the coal from caking in large pieces.
2. The apparatus is to be provided with the fewest projections or appendages possible, consistent with its proper working, in order to prevent semi-coal from accumulating and hanging on.
3. For evident economic purposes, the operations of feeding and discharging must be continuous.
4. Easy access to the inside of all parts of the retort throughout its entire length without stopping the working is necessary for inspecting and cleaning when necessary.
5. The possibility of easy regulation to suit different kinds of coal.

Because of the intention to use coking coal for the production of char, certain types of equipment, especially those employing internal heating and gravity flow, were automatically ruled out. The Hayes retort was the only one found that had been in fairly recent operation in this country and that met most of the requirements outlined above. A pilot plant consisting of one full-sized Hayes retort, with feed bins, conveyors, pulverizers, etc., to make its continuous operation simple and fairly economical was therefore installed. By-product recovery equipment was installed and at the end of the full series of tests several runs were made to determine the nature and yields of by-products.

Retort and Equipment

Briefly, the Hayes retort used consisted of a steel drum, 20 in. in diameter and 25 ft. long, supported by rollers at the ends and enclosed in a brick furnace. Both ends are closed by stationary end covers with packed joints; at the feed end, coal is fed through a variable-speed screw feeder; at the discharge end, the hot char drops into an enclosed, screw-type quenching conveyor, in which it is cooled with water sprays and from which it is discharged through an ejector with rotating blades. Gas is discharged from the top of the housing at the char-discharge end of the retort.

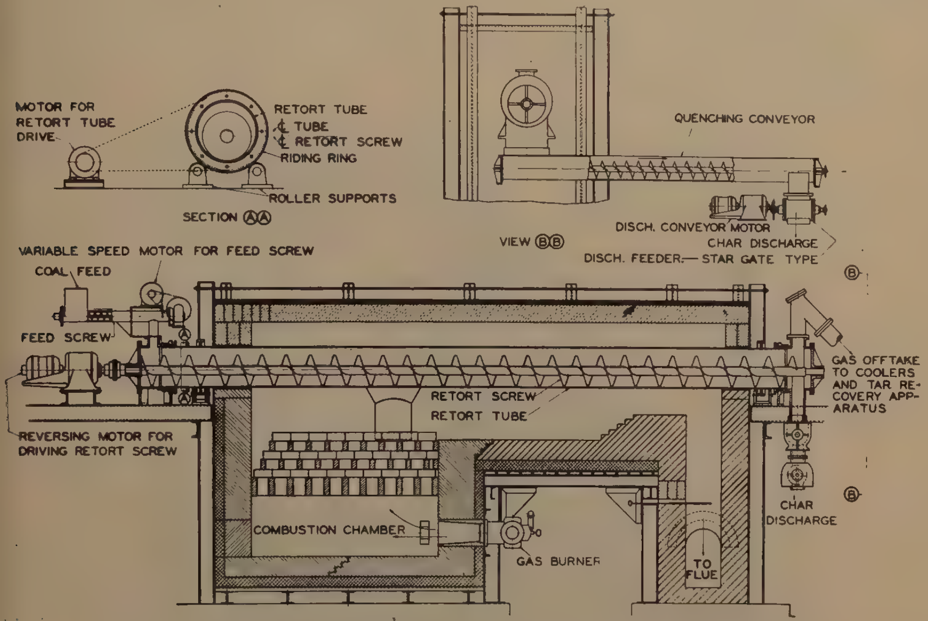

FIG. 1.—HAYES RETORT FOR CARBONIZING COAL AT LOW TEMPERATURES.

The charge is moved through the retort by means of a 16-in. screw extending the full length of the retort, and this screw is driven by a reversing motor, moving the charge forward and then backward for a shorter distance through the retort. This oscillating screw serves a number of purposes. The charge is kept well agitated in the retort; the length of time the charge is held in the retort can be varied by changing the time of forward and backward rotation of the screw and the wiping action of the screw against the walls of the retort delays the accumulation of a hard carbon deposit on the inside of the retort.

The retort furnace was heated by the combustion of pre-mixed air and coke-oven gas, with automatic temperature control. A number of thermocouples, at various points throughout the retort setting, gave a fairly comprehensive picture of the temperature conditions on the outside of the retort. Unfortunately, no provision could be made in this type of retort for the measurement of inside temperatures, therefore there is no information as to the actual temperatures reached by the coal particles themselves in their passage through the retort. A cross section of the Hayes retort installed at the Colorado Fuel and Iron Corporation's coke plant, in which these tests were made, is shown in Fig. 1. The photograph in Fig. 2 shows the installation.

The rest of the equipment consists of bins for the storage of coal and char and conveyors for feeding the coal to the retort feed bin, for carrying the char to the hammer mill and char bin, for mixing coal and char, and for delivery of the mixture to conveyors to the ovens. The by-product equipment consists of a hydraulic main equipped with liquor sprays, a tubular cooler, a coke-filled tar scrubber, decanter, exhauster and liquor-circulating pumps.

PREPARING THE COAL

As mentioned, this work was begun with the idea of producing char from the same coking coal that was to be used in preparing the final blend of coal and char for charging the ovens. It did not take long to find that the statement in West's patent that the material as discharged is "a fused and gummy mass" was no exaggeration. It was gummy and sticky and was not controllable by change in the rate of feed or velocity through the retort. The only thing that could be done to make it work satisfactorily was to raise the temperature of the retort, thereby reducing the volatile of the char—but then the greater part of the value of the char toward improving the coke structure was gone.

Since it appeared that suitable char could not be produced from this straight coking coal, the problem then became one of finding some means of preparing the coal so that it would be satisfactory. A number of possibilities were evident. The coal could be preheated or oxidized, and some of its coking properties destroyed, before it was charged into the retort. The coal could be mixed with some inert material such as coke breeze, or some of the fine char could be returned to the coal and recycled.

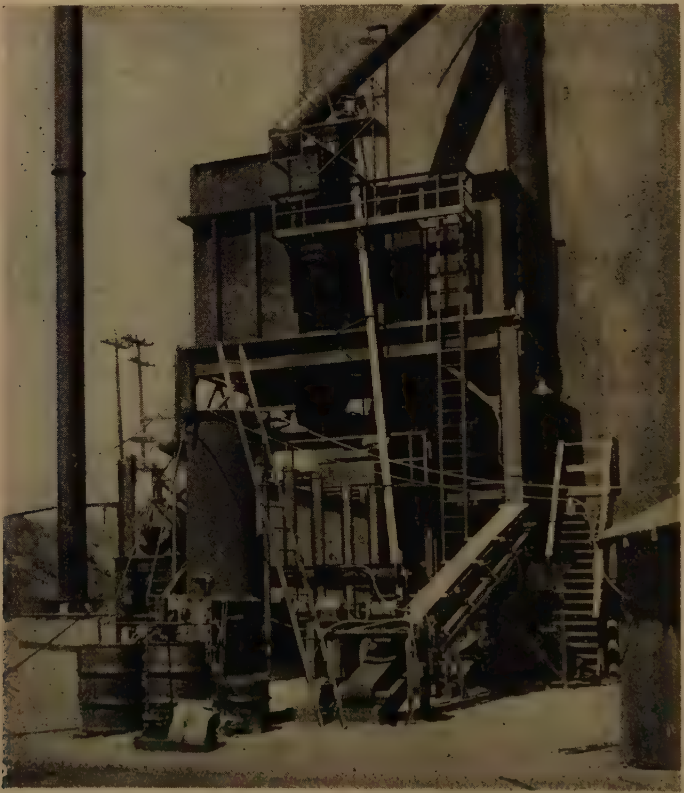

FIG. 2.—HAYES PROCESS FOR COAL CARBONIZATION, ALLIS-CHALMERS MANUFACTURING CO. PILOT PLANT OF COLORADO FUEL AND IRON COMPANY.

These were tried, and they did improve the operation of the retort, but they reduced the value of the char from the angle of final coke improvement.

Then the addition of a high-volatile, 12 to 14 per cent oxygen, noncoking coal to the coking coal before the retort was tried. The addition of 10 per cent of this noncoking coal showed distinct evidence of improvement in the quality of the desired char. The amount was then gradually increased to 50 per cent. As the percentage of noncoking coal was increased, the ease of production of suitable char also improved. At a 50–50 mixture, the operating difficulties of obtaining suitable char had practically disappeared, and the quality of the final coke when this char was used as an admixture was considerably better than that produced with char from straight coking coal.

Since the char produced from this 50–50 blend still possessed distinct coking tendencies, and especially since the quality of the coke made from this char approached closely the quality of coke made in the east, the pilot plant was placed on an operating basis, making and storing this improved coke for a period of one year, so that a blast-furnace test run could be made with it.

Unfortunately not enough coke was accumulated for a blast-furnace run sufficiently extended to yield conclusive evidence as to the value of the coke. The run, however, was of sufficient length to give very decided indications of the improved quality of the coke.

With the completion of the blast-furnace test run the first part of the research problem dealing with the possibility of producing a satisfactory coke for blast-furnace use was completed.

Selection of Coal

The second part of the program consisted of investigations into more of the technical and theoretical considerations connected with the use of char. Four major points were investigated, which are worth some elaboration.

First is the matter of selection of the coal from which the char is to be made. As mentioned, the work was started and carried through the first phase of the program on the assumption that the char should be made from the same high-volatile coal that would later be used in the mixture, and that the char should not be devolatilized to an extent that would completely destroy its coking properties.

The use of the noncoking coal mixed with coking coal in the retort feed was intended primarily to make possible the satisfactory action of the retort. In these later tests the percentage of the noncoking coal was increased to 75 and finally to 100. At 100 per cent the operation of the retort was made practically foolproof, since there was no longer any stickiness of the charge to be contended with at any point. Further, and more important, the quality of the final coke made from the char from this noncoking coal was better than that of the coke with char from the mixture of coking and noncoking coals, and just as good as could be secured by the use of Pocahontas coal.

Except for its appearance, which approached to some degree a pebbly or agglomerated structure rather than the true fused structure of high-quality coke, the coke was of as high a quality as could be desired.

However, this condition was not found to be true with all available noncoking coals, for certain of these, especially low-rank sub-bituminous coals and lignites did not give comparable results. Also, certain local semianthracites of volatile about the same as that of the char, were almost worthless. Any attempt to explain this condition would be largely based on guesswork.

The coking coal and the noncoking coal successfully used were secured from the same section of the state and there may be some relationship between them. Considerable variation in quality of coal is often found in the same mine and in the same vein; it is possible that the two coals may have had the same origin and the present difference in coking properties is due to volcanic or other action. This condition may possibly be found to vary with different coking coals, and this, in turn, may require that the selection of a coal for the preparation of char must be made a matter of individual study for each high-volatile coal used.

AMOUNT OF CHAR NECESSARY

Second, the proper amount of char to be added to secure the best quality of coke must be determined. In this regard the use of char will be found to differ somewhat from Pocahontas. With the latter coal, the percentage in the mixture can be varied over quite a range and the higher the percentage of low-volatile coal, until too great an expansion is reached, the better the coke. With char, however, the quantity that may be used is limited.

Considerable improvement in coke structures was caused by the addition of as little as 10 per cent char, and this improvement continued up to 25 per cent char. Above 30 per cent there was a tendency for the coke to take on a decidedly pebbly or agglomerated appearance and to begin to lose strength. When the proper quantity of char, finely pulverized, is used, the physical properties of the coke do not differ greatly from those secured with the use of Pocahontas. In general, and within certain limits, about 5 per cent more char than Pocahontas is required to give comparable results. Our experience, which has been duplicated in tests at some of the eastern plants, has been that 25 per cent char will give coke quite similar to that which can be secured with the use of 20 per cent Pocahontas.

VOLATILE IN CHAR

Third, the percentage of volatile to be left in the char is fairly critical. Colorado coking coal, devolatilized to 15 per cent volatile matter, has lost all of its coking properties and has little value as a low-volatile substitute. The noncoking coal, devolatilized to 15 per cent, has somewhat more value for this purpose. A local semianthracite of 15 per cent volatile has almost no value. The coking coal with volatile content reduced to between 17 and 18 per cent makes the best quality of final char obtainable from that coal.

To secure the best effect with the use of char from noncoking coal, the remaining volatile must be between 18.5 and 19.5 per cent.

It follows that the percentage of volatile left in the char is critical and that this will vary, depending upon the coal from which the char is made.

TEMPERATURE OF DEVOLATILIZATION

Fourth, the temperature at which devolatilization of the char takes place—in other terms, the rate of devolatilization of the coal—appears to be important. In general, the faster the coal is devolatilized, the more valuable, from the angle of improvement in coke quality, is the char in the final mixture. This factor, also, is one for which we are not able to offer an explanation. Chemically, two chars, devolatilized to the same extent from the same coal but at different temperatures, appear similar. Under a microscope, however, a distinct difference in structure can be noted. The char produced at a lower temperature from a noncoking coal appears of much the same structure as the original coal, but when made at a higher temperature the char has an exploded or "popcorn" appearance.

PHYSICAL FUEL VALUE

In order to set up a means of comparison between the various cokes as produced and tested, the standard A.S.T.M. methods for screen analysis, shatter, tumbler and porosity were adopted. In the shatter test, a variation was made, in that the samples for test were prepared from the screen-size fractions and cumulated in the proportion of these various fractions. By this slight change it was found possible to reduce to a considerable extent the variation usually found between companion tests. A cumulative value for these four physical tests was secured by using the formula as presented by R. W. Campbell some years ago.* To arrive at this value the percentage of material in the screen test plus 2 in. (less a debit for material over 4 in.) is given a weight of 10 per cent; the percentage of cells is given a weight of 10 per cent; the shatter results, expressed in percentage

TABLE 1.—*Coke Improvement Tests*

Description or Source of Sample	Screen, +2 In.	Shatter, +2 In.	Tumbler, +1 In.	Porosity, Per Cent	PFV
Standard coke....	74.4 100	60.6 100	53.7 100	50.5 100	56.5 100
Regular coke.....	78.2 105	64.5 106	28.8 54	46.5 92	41.7 74
Addition of coke breeze, 2.5 per cent...........	74.7 100	71.3 118	23.1 43	47.9 95	39.1 69
5 per cent....	71.3 96	73.8 122	13.5 25	49.3 98	32.4 57
Addition of petroleum breeze, 10 per cent........	87.2 117	73.1 121	37.9 71	44.8 89	50.0 88
20 per cent...	88.2 119	78.8 130	42.0 78	41.1 81	53.6 95
Bulk density: 55 lb. per cu. ft....	76.2 105	64.5 106	28.8 54	46.5 92	41.7 74
57.5 lb. per cu. ft......	76.6 103	61.3 101	30.7 57	44.5 88	42.0 74
60 lb. per cu. ft..........	76.1 102	62.0 102	30.7 57	43.6 86	42.2 75
62.5 lb. per cu. ft......	74.9 101	60.4 100	31.8 59	42.3 84	42.4 75
Pulverization: 70 per cent − ⅛-in.............	78.2 105	64.5 106	28.8 54	46.5 92	41.7 74
80 per cent − ⅛-in....	82.6 111	71.3 118	29.7 55	47.0 93	44.0 78
100 per cent − ⅛-in....	79.7 107	73.3 121	31.2 58	48.9 97	45.8 81
Coking time: 20 hr.	83.2 112	62.3 103	25.8 48	47.9 95	40.4 72
24 hr.........	78.2 105	64.5 106	28.8 54	46.5 92	41.7 74
Addition of 20 per cent Pocahontas	79.8 107	81.3 134	55.4 103	50.0 99	62.2 110
Addition of 20 per cent Howe coal.	87.2 117	79.7 132	47.2 88	49.6 98	55.8 99

TABLE 2.—*Various Char Effects*

Description or Source of Sample	Screen, +2 In.	Shatter, +2 In.	Tumbler, +1 In.	Porosity, Per Cent	PFV
Standard coke....	74.4 100	60.6 100	53.7 100	50.5 100	56.5 100
Regular coke.....	78.2 105	64.5 106	28.8 54	46.5 92	41.7 74
Coal used for char production: 90 per cent coking.	63.3 85	64.3 106	53.1 99	44.1 87	55.3 98
80 per cent coking.....	63.1 85	66.0 109	54.4 101	44.3 88	55.9 99
70 per cent coking.....	67.7 91	70.9 117	53.3 99	45.1 89	57.2 101
60 per cent coking.....	72.8 98	70.7 117	54.0 101	44.8 89	57.9 102
50 per cent coking.....	69.4 93	68.9 114	55.4 103	44.8 89	58.3 103
100 per cent noncoking..	73.6 99	75.3 124	57.5 107	45.2 90	61.1 108
Volatile content of char: 17.4 per cent...........	68.2 92	61.3 101	37.4 70	46.9 93	45.0 80
19.0 per cent..	73.2 98	68.5 113	49.2 92	49.2 97	54.8 97
20.6 per cent..	79.5 107	71.4 118	43.9 82	46.4 92	51.1 90
Char, percentage in final mixture: 0.	78.2 105	64.5 106	28.8 54	46.5 92	41.7 74
10...........	81.2 109	62.9 104	46.7 87	42.5 84	51.0 90
15...........	84.4 113	59.0 97	46.0 86	43.3 86	51.0 90
20...........	81.4 109	59.8 99	47.3 88	43.3 86	52.2 92
25...........	71.0 95	67.6 112	50.3 94	44.3 88	54.1 96
Char, temperature of production: 950°F..........	85.8 115	61.9 102	38.6 72	43.2 86	46.0 81
1450°F.......	87.5 118	66.5 110	45.2 84	42.3 84	51.9 92

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* Paper read Oct. 16, 1936 before the joint meeting of Chicago District and Eastern States Blast Furnace and Coke Oven Association. The use of the Campbell formula for computing the Physical Fuel Value is of importance primarily because it has been tied into blast-furnace performance in this paper. Other formulas for computing the P.F.V. of coke that have been tied into furnace operation will be of equal value.

remaining on a 2-in. screen, is given a weight of 20 per cent, and the tumbler results, in terms of the percentage remaining on a 1-in. screen, is given a weight of 60 per cent. This cumulated value is given the name of Physical Fuel Value, and presents a very simple way of cumulating the results and of comparing the physical values of different cokes.

In order to have some standard of coke quality at which to aim, two samples of coke were secured from eastern plants, one from the Chicago and one from the Pittsburgh district. Both of these were made in ovens between 16 and 18 in. wide, from mixtures of high-volatile coal with 20 per cent Pocahontas, but at somewhat faster coking times than usually are used at Colorado. These were tested in the laboratory and the results set up as a standard.

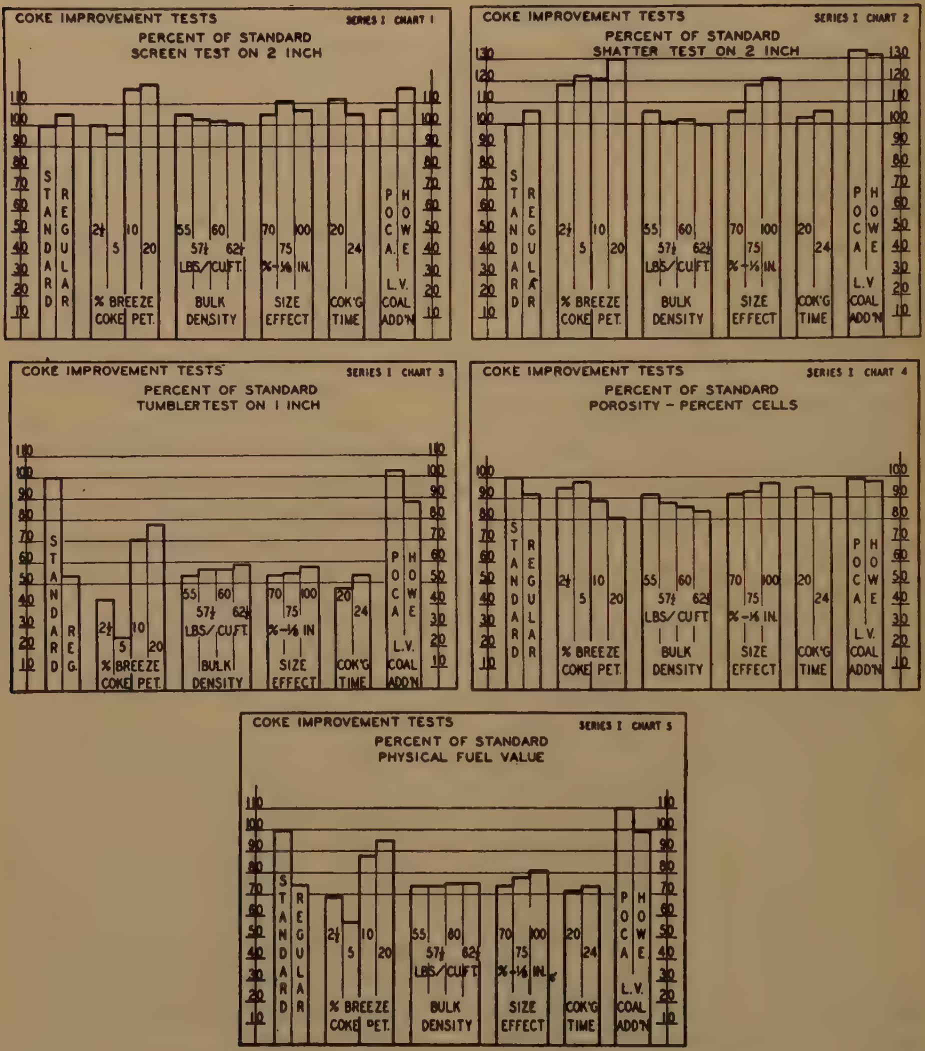

FIG. 3.—RESULTS OF EFFORT TO IMPROVE BLAST-FURNACE COKE.

Against this standard there was set up the results of similar tests on average Colorado Fuel and Iron Corporation blast-furnace coke, then the results secured through testing cokes made under the investigated methods of improving this coke. These are shown in the accompanying tables and charts. In Tables 1 and 2 the results of each test are shown in two ways: The upper figure in each case is the actual test result; the lower figure is an index figure, representing the percentage of accepted standard of that test result. For example, if the accepted standard shatter is 60.6, or an index of 100, a test result showing 40.4 would represent a value of 66.67 per cent of standard; in whole numbers, it would be given an index of 67. These indexes only, and not the actual test results, are used in preparing the charts.

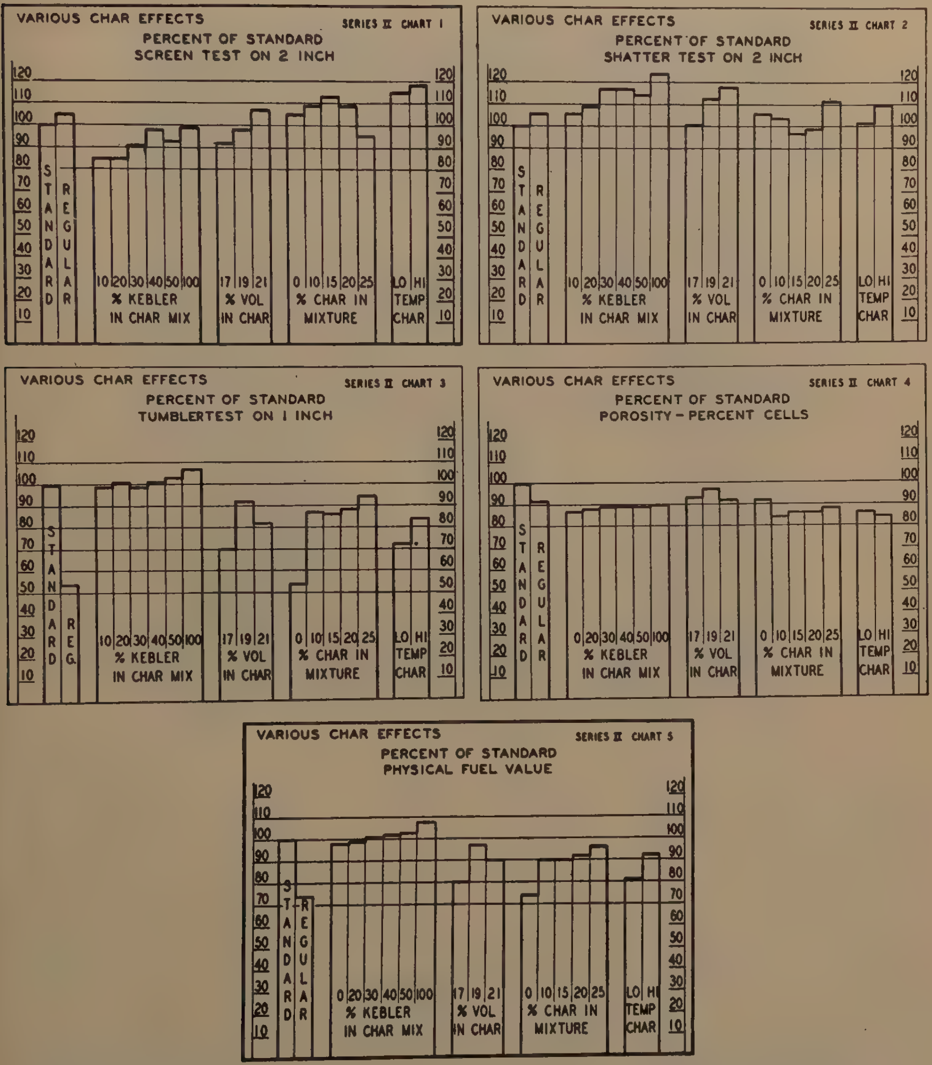

FIG. 4.—EFFECT OF CHARS ON PHYSICAL PROPERTIES OF COKE PRODUCED FROM THEM.

RESULTS OF TESTS

Two separate series of these charts have been prepared (Figs. 3 and 4). The first series of five charts (Fig. 3) shows a few of the results of the work that was first performed in an effort to improve the blast-furnace coke. In this series is shown the effect of the addition of coke breeze and of petroleum-coke breeze, of variations in bulk density, of variations in degree of pulverization, the effect of variation in coking time and the effect of Pocahontas and of Howe coal, the latter being a coal of properties very similar to those of Pocahontas but secured from eastern Oklahoma.

The second series of charts (Fig. 4) shows the effect of various chars on the improvement of the physical properties of coke produced from them. Here is shown, along with the standard and regular cokes for comparison, the effect of a change in the kind of coal used for the making of the char. In this case a mixture at 90 per cent coking coal and 10 per cent noncoking coal is increased in increments of 10 per cent of noncoking coal to a 50–50 mixture, and with a char made from 100 per cent noncoking coal shown for comparison. Here also is shown the results of variation in the volatile content of the char, of the percentage of char in the final mixture as charged into the ovens, and of the furnace temperature at which the char is produced.

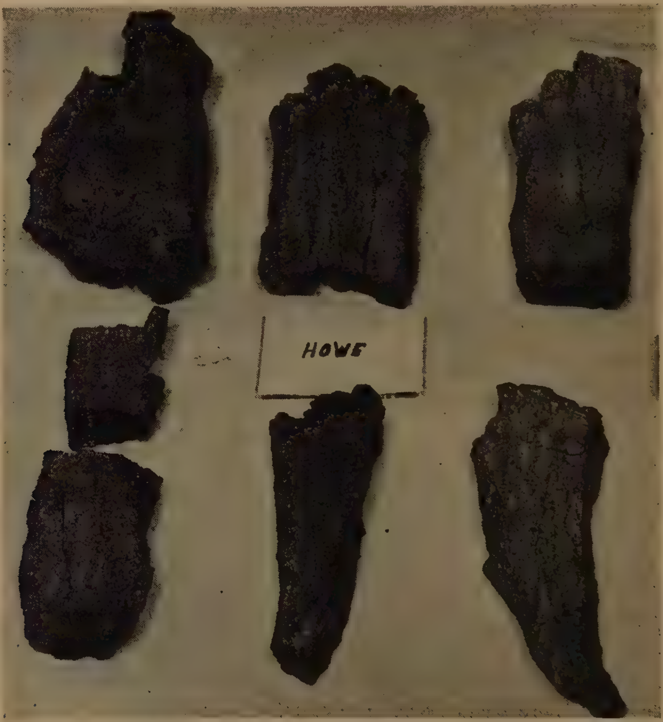

FIG. 5.—APPEARANCE OF COKE REGULARLY PRODUCED FROM 100 PER CENT HIGH-VOLATILE COAL.

The results as listed in the tables and shown on these charts were taken more or less at random from a mass of data accumulated during the three years of intensive experimental work; in most cases, they represent an average of a large number of test runs.

On the first chart, representing the percentage of plus 2-in. in the A.S.T.M. screen test, the regular coke, and, with only one exception, all of the test cokes, show an index above standard. Probably this is because the coke is regularly produced at 24 hr. coking time and the standard cokes were produced at faster rates. No complaint is made as to the size of the regular coke.

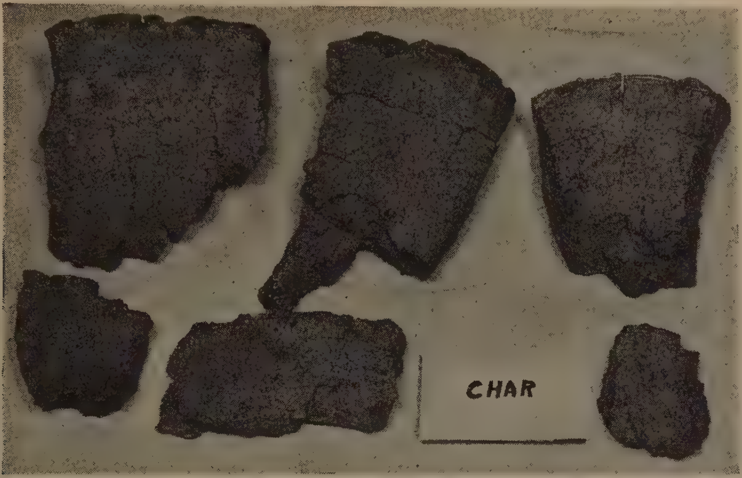

FIG. 6.—APPEARANCE OF COKE PRODUCED WITH MIXTURES OF HIGH-VOLATILE COKING COAL AND LOW-VOLATILE COAL AND CHAR.

On the second chart is shown the percentage of standard shatter; that is, the amount of coke remaining on the 2-in. screen after the shatter test as compared with the standard coke. Here it will be noted that the regular coke and all of the test cokes have shown an index above standard. One of the peculiarities of the Colorado coals is that they yield a coke of fairly high shatter value but, as will be seen on the next chart, of low tumbler index. No fault is found with the shatter strength of the regular coke. The addition of coke breeze raised the shatter index considerably and the addition of petroleum-coke breeze did the same.

Changing the bulk density of the charge had but little effect on the shatter index. Increasing the fineness of pulverization improved the shatter. Change in coking time had little effect. The addition of Pocahontas coal, or the Oklahoma low-volatile coal, which is quite similar, had the greatest effect in increasing the shatter index.

On the third chart is shown the chief objectionable feature in the regular coke; that is, its low resistance to abrasion. This regular coke has only about one half the resistance of the standard. The effect of finely pulverized coke breeze was harmful but petroleum breeze improved the product slightly. Increasing the bulk density of the charge caused a very slight improvement, and so did finer crushing. With a coking time faster than the 24-hr. rate, which has been set up as the normal operating rate, the resistance to abrasion drops off.

Pocahontas coal, in the proportion of 20 per cent to 80 per cent of the high-volatile coal, gives the greatest improvement and shows that a high-grade coke can be made from the Colorado coals. The Oklahoma coal, similar to Pocahontas, gave considerable improvement, although not as great as was secured by the use of Pocahontas.

Chart 4 shows the porosity of the various test cokes as compared with standard. In no case was it possible to reach the standard of eastern cokes, Pocahontas and Howe coming closest. Coke breeze had a beneficial effect; petroleum coke reduced the porosity. As would be expected, increasing the bulk density of the charge decreased the percentage of cells. Finer pulverization raised it. A faster coking time improves it somewhat. Pocahontas and Howe bring it up to about standard.

On the fifth chart is shown the physical fuel values for these cokes. Here again it will be noted that the standard of eastern cokes is met only by the use of Pocahontas or Howe coal in the mixture. The use of 20 per cent petroleum breeze comes close; but only a comparatively small amount of that material is available. With the exception of coke breeze, addition of which shows an actual reduction in quality, practically all of these tests have indicated some slight improvement. Needless to say, where any method of improvement, even though slight, was found practical, this method has been adopted in regular practice, and the regular coke at present is somewhat better than that shown on this chart.

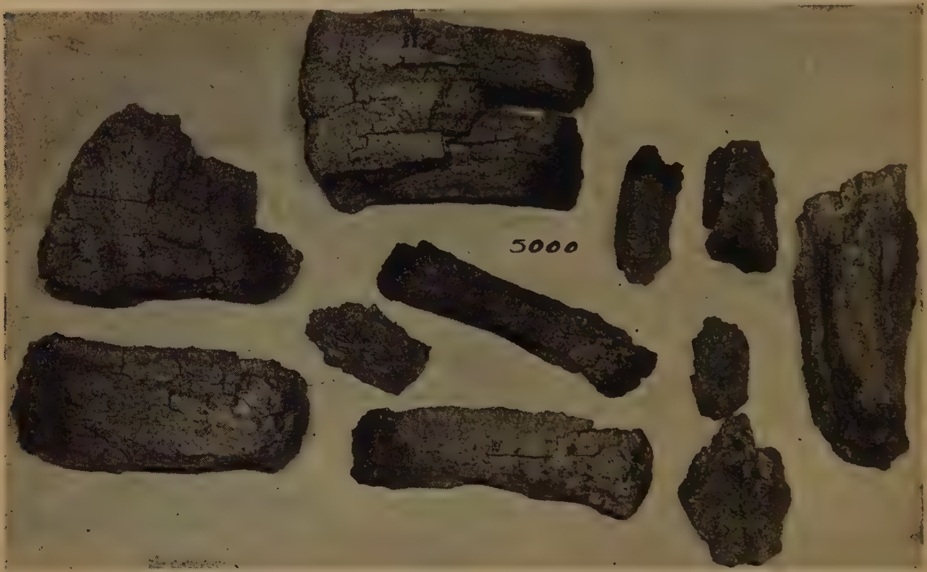

Fig. 7.—Regular production, 5000 grade of coke, showing cross fracturing and fingering tendency.

The second series of five charts (Fig. 4) shows the results of the four physical tests and the physical fuel value on a number of samples of coke made with the use of char, the samples being selected to show the effect of the various properties of char on the quality of coke so made and as discussed above. The first chart shows the percentage retained on a 2-in. screen in the standard screen size test. The first group of tests shows the effect of increasing the percentage of noncoking coal in the mixture to the char retort.

As has been mentioned, this series of tests was first conducted up to 50 per cent of the noncoking coal only, and this mixture was accepted as satisfactory for all further work. The 100 per cent noncoking coal was tried after the completion of the greater part of the other test work. The results will show why it was decided that this is the proper coal to use in preparing char. On this chart not much can be said for the effect of the various char conditions on size of coke; all of the chars tested come within a fair range of standard.

TABLE 3.—*Average Analysis of Kebler and Frederick Coals*

Name of coal	Kebler		Frederick	
Classification	Noncoking		Heavy coking	
Coal preparation	Raw	Washed[a]	Raw	Washed[a]
Proximate analysis, per cent volatile matter	38.5	40.0	30.6	32.0
Fixed Carbon	48.4	51.5	55.3	59.6
Ash	13.1	8.5	14.1	8.4
Moisture		11.0		12.0
Ultimate analysis, per cent: carbon		78.27		85.44
Hydrogen		5.67		5.64
Oxygen		13.96		6.40
Nitrogen		1.41		1.92
Sulphur		.69		.60
Screen: +½-in.		1.2		1.7
¼ – ½-in.		7.5		10.8
⅛ – ¼-in.		23.2		17.9
–⅛-in.		68.1		70.6

COMPLETE ANALYSIS OF CHAR FROM KEBLER COAL

Proximate analysis, per cent volatile matter	19.1	Screen: +½-in.	0.0
Fixed carbon	70.4	¼ – ½-in.	5.7
Ash	10.5	⅛ – ¼-in.	22.7
		–⅛-in.	71.6

[a] Washed coal used in preparation of char and in final mixture.

On chart 2 is shown the shatter-test results. Here the outstanding value is the char made from 100 per cent noncoking coal. All the results are high as compared with the standard coke and there is little to discuss. It might be mentioned, however, that of the various groups of tests shown here, the 50–50 mixture, 19.0 per cent volatile, 25 per cent char in final mixture and furnace temperature of 1450°F. were accepted as optimum conditions for the preparation of the best char.

Chart 3 shows the tumbler-test results. Since this property of the coke has been given the greatest weight in the determination of the physical value of coke for the blast furnace, we are primarily interested in these figures. We note in the first group of tests that the tumbler index gradually shows an increase up to the 50–50 mixture, that the 19 per cent volatile char is better than either the lower or higher value, that 25 per cent char makes a better final coke than does any smaller amount and that the higher temperature of preparation makes the best final product.

TABLE 4.—*Analysis of Principal Cokes*
PER CENT

Kind of coke	Standard	Regular	25 Per Cent Kebler Char Coke
Porosity	50.5	46.5	45.5
SCREEN TEST			
Cum. on 5		4.4	
4	5.4	8.3	5.6
3	27.7	40.8	37.2
2	74.4	79.6	85.4
1½	92.0	92.7	96.6
1	98.5	97.4	99.5
½	99.3	98.7	100.0
–½	100.0	100.0	
SHATTER TEST			
Cum. on 4		1.6	5.5
3	6.7	17.8	26.6
2	60.6	64.5	79.1
1½	86.9	83.7	92.4
1	96.8	92.4	98.2
½	98.7	97.2	99.3
–½	100.0	100.0	100.0
TUMBLER TEST			
Cum. on 2	4.1	0.4	6.3
1½	27.1	7.3	28.0
1	52.7	28.8	56.0
½	65.6	53.8	65.6
¼	67.3	63.4	66.7
–¼	100.0	100.0	100.0
Coke P.F.V.	56.5	41.7	61.1[a]
SCREEN TEST OF KEBLER CHAR AND ANALYSIS OF FRACTIONS			
Size fractions, in.	+¼	¼–⅛	–⅛
Feed coal, per cent fraction	3.8	23.3	72.3
Feed coal, per cent volatile	37.4	40.6	40.2
Char, per cent fraction	6.4	21.0	72.3
Char, per cent volatile	21.7	22.7	22.4
Percentage reduction in volatile	42.0	44.1	44.3

[a] Best char coke made 23 hr. coking time.

On chart 4 is shown the porosity of these same samples. It will be recalled that a porosity of about 50 per cent was shown as secured by the mixture of Pocahontas or Howe coal with the local high-volatile coal. With char this advantage is not secured and the char coke probably will average a little denser than the regular run of coke. While this might tend to impair its combustibility in the furnace to some extent, it is felt that this slight disadvantage will be more than offset by the increase in its resistance to abrasion.

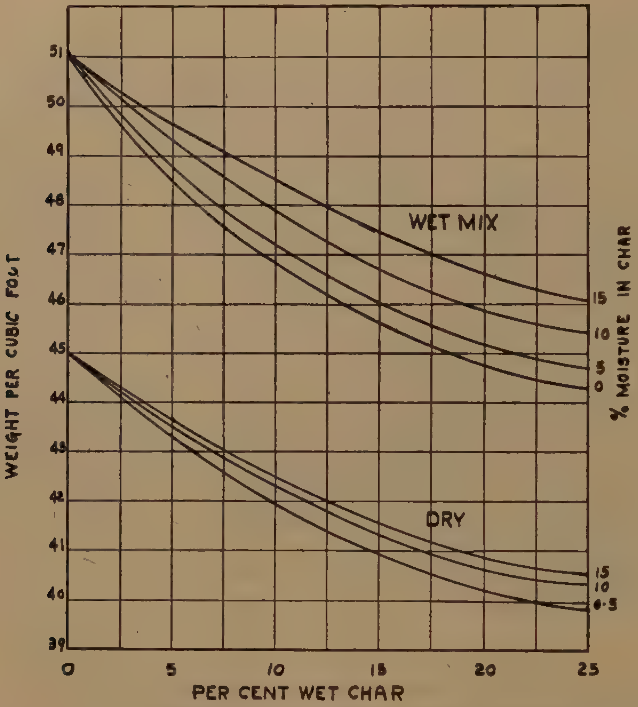

Fig. 8.—Effect of char on bulk density of mixes.

Experimental work done with char crushed to pass ⅛ in. and then dried. Moisture added while char was hot. Coal used was taken from coal bin and contained about 12.0 per cent moisture. Bulk density of dry char about 41.6 lb. per cu. foot.

Chart 5 shows the physical fuel value of these char cokes. The char first accepted as satisfactory did not give a coke of P.F.V. equal to standard. However, it was felt that the coke was sufficiently close to standard to warrant the temporary stopping of the research work and the placing of the retort on a production basis for a blast-furnace run. The later results shown for 100 per cent noncoking coal show the reason for the special interest in this type of char.

Characteristics of Coal and Char

The actual appearance of the coke as regularly produced from 100 per cent high-volatile coal and of coke produced with mixtures of this coking coal with low-volatile coal and char is shown on Figs. 5 and 6. The 5000 grade of coke, the regular production, shows plainly the cross fracturing and fingering tendency of this coke (Fig. 7). This tendency is largely eliminated in the coke made from the mixture with low-volatile coal. Some cross fracturing is noticeable in the char coke, but the fingering tendency is largely eliminated and the tendency is toward the formation of smaller but blockier pieces of coke. Since the regular coke is somewhat larger than the standard, some decrease in size of individual pieces is not objectionable.

At this point it might be interesting to learn something about the coals, char and by-products discussed herein.

In Table 3 is an average analysis of the Frederick coking and the Kebler noncoking coals.

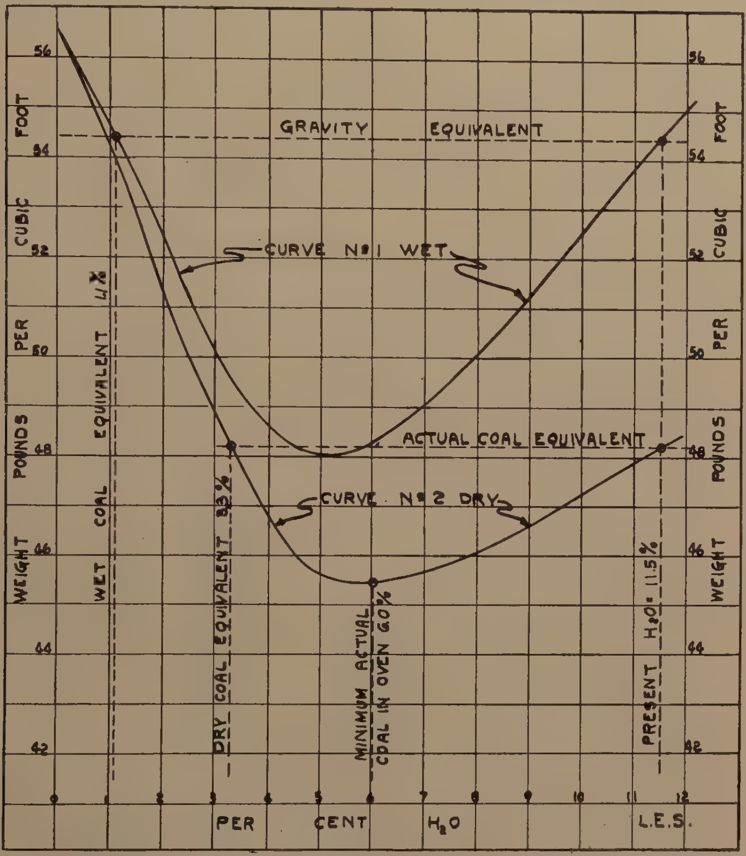

FIG. 9.—EFFECT OF MOISTURE ON UNIT WEIGHT OF COAL.

A complete average analysis of the char made from 100 per cent Kebler coal is shown also in Table 3. The sizing test was made on material from the retort. This was reduced by hammer mill, so that 99.5 per cent passed through a ⅛-in. screen before mixing with coking coal.

A physical analysis of the three principal blast-furnace cokes under discussion are given in Table 4.

The screen test of Kebler char and an analysis of various fractions at the end of Table 4 is of considerable interest, since it shows practically equal devolatilization of all particles of coals regardless of their size.

A study of the bulk density of char and of coal-char mixtures and its effect upon oven capacity is of interest. Fig. 8 shows the effect of char on bulk density of mixes. Here it is shown that increased percentages of char of varying moisture content mixed with wet coking coal will cause a decrease in the bulk density of the mixture up to 25 per cent of char addition. This indicates that when using 25 per cent char of 10 per cent moisture content, bulk density of the charge will be reduced from 51.1 lb. to 45.4 lb., or about 11 per cent. Hence the amount of coal-char mixture that can be charged into an oven will be decreased by this amount. Considering now, the dry coal portion of this total charge, we find that this will be reduced from 44.9 to 40.3 lb., a reduction of 10.2 per cent. But we have the increased yield per oven to offset this. Based on a mixture charged of 90 per cent Frederick at 32.0 per cent volatile and 10 per cent Kebler at 40.0 volatile, we would have a volatile content in the mixture charged of 32.8 per cent, an indicated coke yield of 67.2 per cent. With a mixture of 75 per cent Frederick and 25 per cent char at 19 per cent volatile, the volatile content of the mixture would be 28.8, an indicated coke yield of 71.2 per cent. This indicates an increase in coke yield per oven of

$$(71.2 - 67.2) \div 67.2 = 6.0 \text{ per cent},$$

leaving a net loss in coke per oven pushed of 4.2 per cent. However, if coking time can be reduced from 24 to 20 hr., ovens per day would be increased 20 per cent. Thus we have a net gain in coke capacity of about 15 per cent. However, coke made at 20 hr. coking time with char is slightly inferior to that made at 24 hours.

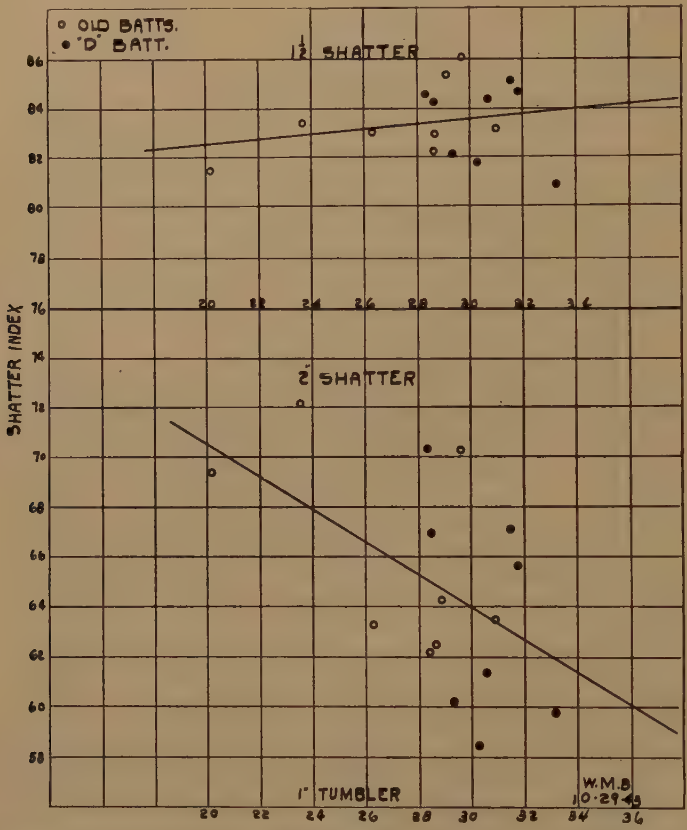

FIG. 10.—CORRELATION OF SHATTER AND TUMBLER TESTS ON 5000 BLAST-FURNACE COKE, JUNE AND JULY 1939.

Some discussion as to the cause underlying the decreasing of bulk density through the addition of char may be in order. The bulk density of coal, as shown on Fig. 9, decreases to a minimum at about 5 per cent water and then increases at a slower rate. The amount of dry coal in the wet coal decreases to about 6 per cent water content, and then slowly increases. From 0 to 5 or 6 per cent moisture addition, the coal apparently becomes "fluffy" and the tendency toward packing is decreased. As the moisture exceeds the 5 or 6 per cent mark, this tendency toward fluffiness disappears and the coal begins to pack. Naturally, these curves vary, depending upon the specific gravity of the coal, pulverization of the coal and the height from which the coal is dropped into the test box. With char, the action is entirely different. Dry char is spongy in nature and will absorb water up to about 5 per cent without any change whatever in its apparent volume and without any change in its packing tendency. Therefore the addition of any amount of char up to 25 per cent in this moisture range will cause a decrease in the bulk density of the mixture. Above 5 per cent moisture in char, there is a slight packing tendency, so that with char of this nature added to coal there will be a somewhat higher bulk density at a corresponding

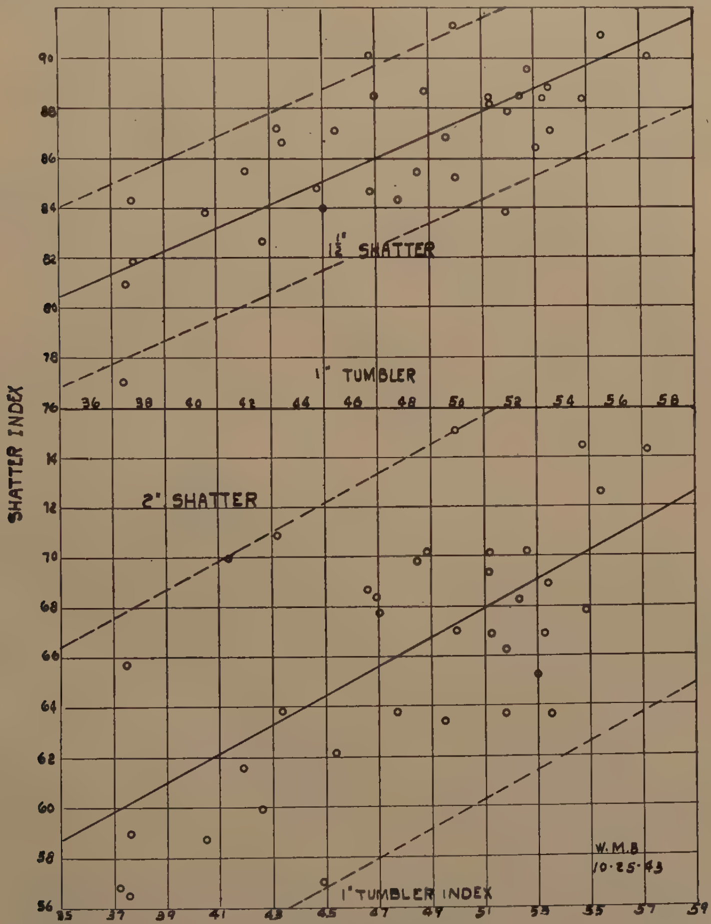

FIG. 11.—CORRELATION OF SHATTER AND TUMBLER TESTS ON CHAR COKES. ALL TYPES, BOX AND OVEN TESTS.

percentage of char, and slightly higher dry material content, but apparently never approaching the bulk density of straight coal.

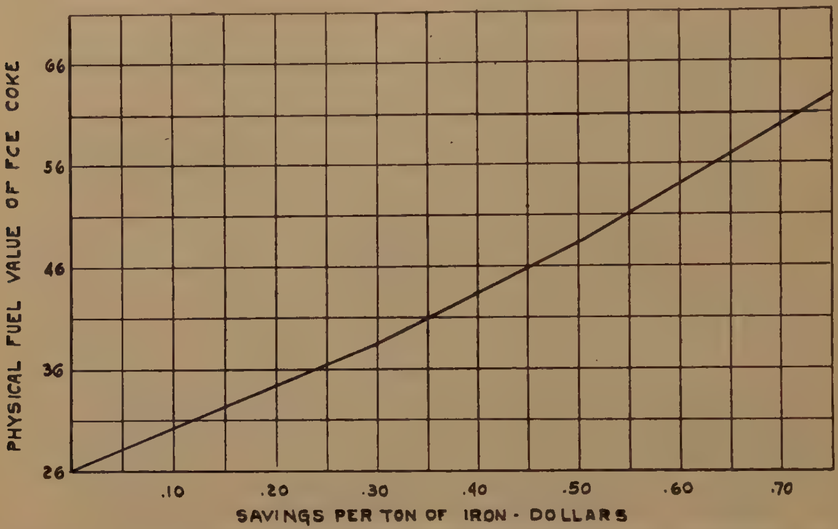

FIG. 12.—SAVINGS PER TON OF IRON BASED ON PHYSICAL FUEL VALUE OF FURNACE COKE.

RELATIONSHIP BETWEEN SHATTER TEST AND TUMBLER-TEST INDEXES

In the preliminary work prior to the intensive research program started in 1938, it was found that in general any improvement in the shatter index of the coke was accompanied by a decrease in the tumbler index. This is shown by comparing samples selected at random from the regular 5000 coke and as shown on Fig. 10. On the lower half of this figure, the 2-in. shatter index is plotted against the 1-in. tumbler index and the trend is definitely shown. When comparing the 1½-in. shatter index, the trend is opposite but considerably less than is secured by plotting similar data as shown on Fig. 11.* This indicates the value of char as an improvement agent for the coke over any other means tried except the use of Pocahontas, Howe coal, or petroleum breeze, and the latter material is not available for use. The data on Fig. 11 also indicate the value of adopting

TABLE 5.—*Data on Runs with Kebler Coal and Regular Mixture*

Coal used	100 per cent Kebler	Regular mixture
Coke or char made in	Hayes retort	Koppers ovens
Tar yield, gal. per ton, dry basis	38.0	10.5
Specific gravity	1.105	1.180
Fluid point	70°C.	40°C.
Distillation: To 210°C.	2.4	0.7
235°C.	6.2	4.7
315°C.	23.3	19.0
355°C.	41.1	27.5
370°C.	38.9	31.3
380°C.	43.9	40.5
Residue above 380°C.	55.5	59.2
Tar acids	15.6	4.0
Gas yield, cu. ft. per ton	1,600	11,800
Gross B.t.u. (air free)	818	550
Net B.t.u. (air free)	744	490
SUMMARY OF PRODUCTS RECOVERED, LB. PER TON COAL USED		
Coke or char	1,560	1,476
Tar	348	106
Gas	64	367
Ammonia in liquor	0.4	5.6
Phenol in liquor	2.2	
Soluble tar in liquor	15.5	23.4
Unaccounted for loss	9.9	22.0
Total	2,000	2,000

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* It is emphasized again that the purpose of including these charts showing the relationship between shatter and tumbler tests is to emphasize the fact that with the char used in the coking mixture, the shatter and tumbler indexes show the same general trend but without char the trend of each may be and often is in opposite directions. This seems to be a peculiarity of Colorado coals and is not generally true of Eastern high-volatile coals.

as standard the 1½-in. shatter index rather than the 2-in. index now in general use, because the increased precision of estimate as indicated by the narrow band on the table shows its greater reliability.

In Table 5 are shown the yield and analysis of tar and by-products from low-temperature runs on Kebler coal compared with yields from regular high-volatile mixture (90 per cent Frederick, 10 per cent Kebler) to high-temperature ovens.

EVALUATION OF COKE PRODUCED WITH CHAR

Experimental and development work generally is pursued to produce new or better products or to improve processes for production at an increased rate or lower cost, or both. The experiments reported herein were made to determine whether better metallurgical coke could be produced at permissible costs so as to obtain a greater output of pig iron from blast furnaces. The report would not be complete unless an attempt were made to evaluate the seemingly better coke resulting from the admixture of low-temperature char with high-volatile coal.

In the absence of definite information as to the worth of each point in increased physical fuel value of the coke charged to the blast furnaces in question, it was necessary to assume the "savings per ton of iron based on P.F.V. of furnace coke," as set forth in R. W. Campbell's paper of 1936, "Physical Testing of Coke and Correlation with Furnace Operation."

Table 6 is one method of evaluating the over-all results of the test work. The assumed capacity is 1600 tons per day of coke charged to the blast furnaces. The cost of the coal and the credits for tar, gas, and light oil approximate the average market for these products in the Colorado district. The credits or savings assumed for the increased P.F.V. of the blast-

TABLE 6.—*Comparative Blast-furnace Savings Due to Use of Low-temperature Char as Admixture with Colorado High-volatile Coking Coals*[a]

H.V. Coal-char Mixture	100 % H.V.	90 % H.V., 10 % CH.	75 % H.V., 25 % CH.
Coke required per day, tons	1,600	1,600	1,600
Coke yield, per cent	65	66½	70
Coal or coal-char mix charged, tons	2,460	2,400	2,285
Cost of coal, 2460 × $4.12½	$10,150.00		
Cost of coal-char mix:			
90 % × 2400 × 4.12½ = 8910			
10 % × 2400 × 3.94 = 945		$ 9,855.00	
75 % × 2285 × 4.12½ = 7070			
25 % × 2285 × 3.68 = 2100			$ 9,170.00
Tar yield per ton of feed, gal	10	9½	8¾
Gallons per day	24,600	22,800	20,000
Credit at 6¢ per gal	$ 1,476.00	$ 1,368.00	$ 1,200.00
Gas yield and credit:			
2460 × 11800 at 6¢ per 1000 cu. ft.	$ 1,742.00		
2400 × 11520 at 6¢ per 1000 cu. ft.		$ 1,663.00	
2285 × 11100 at 6¢ per 1000 cu. ft.			$ 1,522.00
Light oil yield and credit:			
2460 × 2 at 8¢ per gal	$ 394.00		
2400 × 2 at 8¢ per gal		$ 384.00	
2285 × 2 at 8¢ per gal			$ 366.00
Physical fuel value of blast-furnace coke	41.7	57.3	61.1
Estimated savings per ton pig iron[b]		0.31	0.37
Estimated daily savings (1600 tons pig iron)[c]		$ 496.00	$ 592.00
Total daily savings and credits—tar, gas, L.O. and P.F.V	$ 3,612.00	$ 3,911.00	$ 3,679.00
Daily fuel charge cost less credits	$ 6,538.00	$ 5,944.00	$ 5,491.00
Daily savings due to use of char in mix		$ 594.00	$ 1,047.00
Yearly savings due to use of char, without depreciation		$217,000.00	$381,000.00
Yearly savings due to use of char, 20 per cent depreciation		$167,000.00	$241,000.00

[a] No penalty is assessed against coke-oven capacity because of lower bulk density of coal-char mix, and no credit is taken for shorter coking time.

[b] See Fig. 10 for basis of this credit.

[c] Based on use of one ton coke per ton pig iron.

furnace coke made from a coal-char mix were taken from Fig. 12. The coke yield from 100 per cent Colorado coal is approximately 65 per cent and from 19 per cent volatile char about 80 per cent. These yields were used in calculating the

TABLE 7.—*Estimated Production Cost of Low-temperature Coke Char by the Hayes Process*

Item	Total	Per Ton Char
Estimated plant cost: 56 retorts, 14 furnaces	$ 700,000	
Capacity, tons:		
Feed coal per day (24 hr.)	790	
Feed coal per year (330 days)	261,000	
Char per year (based on 80 per cent yield)	209,000	
Operating costs:		
Coal at $4.12½ per ton delivered (dry basis)	$1,077,000	
Labor	68,700	
Supervision and chemist	11,000	
Power, 643 net hp. at $0.003 per kw-hr	16,300	
Fuel, equivalent 3 tons per day at $4.12½ per ton	6,800	
Water, 500 gal. per min. at $0.02 per 1000 gal.	4,700	
Repairs and supplies, $0.10 per ton feed, material only	26,100	
A. Total operating costs	$1,210,600	$5.79
Credits:		
Crude tar, 35 gal. per ton; 9,150,000 gal. at 5¢ per gal.	458,000	
B. Total credits	$ 458,000	$2.19
Net cost without fixed charges (A minus B), L.T.	$ 752,600	$3.60
Fixed charges, 2.5 per cent taxes and insurance	$ 15,750	0.08
Net cost at coke plant without depreciation	$ 768,350	$3.68

LABOR AND SUPERVISION

Labor	Men per Shift	Number of Shifts[a]	Man Hours	Hourly Rate	Daily Cost
Retort attendants	2	3	48	0.90	$ 43.20
Recovery-plant attendants	1	3	24	0.90	27.60
Maintenance crew	4	1	32	1.00	32.00
Laborers	3	3	72	0.80	57.60
Foreman	1	3	24	1.25	30.00
Total daily labor cost					$ 184.40
Yearly labor cost					$68,700.00

[a] Labor shifts 8 hours.

TABLE 8.—*Estimated Production Cost of Low-temperature Coke Char by the Hayes Process*

Item	Total	Per Ton Char
Estimated plant cost: 24 retorts, 6 furnaces	$250,000	
Capacity, tons:		
Feed coal per day (24 hr.)	316	
Feed coal per year (330 days)	104,300	
Char per year (based on 80 per cent yield)	83,500	
Operating costs:		
Coal at $4.12½ per ton delivered (dry basis)	$430,000	
Labor	47,500	
Supervision and chemist	7,000	
Power, 292 net hp. at $0.003 per kw-hr	7,400	
Fuel, equivalent 2 tons per day at $4.12½ per ton	2,720	
Water, make-up only, 50 gal. per min. at $0.04 per 1000 gal.	950	
Repairs and Supplies $0.10 per ton feed, material only	10,430	
A. Total operating costs	$506,000	$6.60
Credits:		
Crude tar 35 gal. per ton, 3,655,000 gal. at 5¢ per gal.	$183,000	$2.19
B. Total credits	$183,000	$2.19
Net cost without fixed charges (A minus B), L.T.	$323,000	$3.87
Fixed charges, 2.5 per cent taxes and insurance	$ 6,250	.07
Net cost at coke plant without depreciation	$329,250	$3.94

LABOR AND SUPERVISION

Labor	Men per Shift	Number of Shifts[a]	Man Hours	Hourly Rate	Daily Cost
Retort attendants	1	3	24	1.00	$ 24.00
Recovery-plant attendants	1	3	24	0.90	21.60
Maintenance crew	2	1	16	1.00	16.00
Labor	2	3	48	0.80	38.40
Foremen	1	3	24	1.25	30.00
Total daily labor cost					$ 130.00
Yearly labor cost					$47,500.00

[a] Labor shifts 8 hours.

coke yield from the high-temperature ovens when using various percentages of coal and char mixtures.

To arrive at the cost of a given mix to high-temperature ovens, it was necessary, therefore, to estimate the cost of the char entering into this mix. Tables 7 and 8 indicate how the estimated cost of this char was arrived at; first on the basis of 790 tons per day, which would be the coal requirement to the low-temperature retorts for a 25 per cent char mix, and 316 tons per day, which would be the coal requirement to the low-temperature retorts for a 10 per cent char mix. For the larger amount of char required, the cost per ton of char figured out at $3.68 per ton and for the smaller amount this cost was $3.94 per ton. These costs do not include depreciation, as it was thought best to make up the figures on the basis of yearly savings due to the use of char, without plant depreciation, thereby permitting the reader to determine readily how long it would take a blast-furnace operator to pay for the original investment for making char.

No credit is taken for light oil in the char plants but 2 gal. per ton is assumed for the entire charge to the high-temperature ovens. The amount of gas produced by the char plants is about equal to the amount of gas required to heat the retorts, so that no charge or credit is taken or given to the low-temperature plant for the gas used or produced. A greater percentage of the gas is retained by the char and is available from the high-temperature coking process.

If a low-temperature char plant for producing sufficient char for a 25 per cent mixture would cost $700,000, and if the yearly savings due to the use of this char were $381,000, as indicated, an investment in the char plant could be paid for in approximately one year and ten months, and thereafter the yearly savings should be $381,000. While further investigations and discussions might change these overall figures, there is a very definite indication that under these conditions an investment in a low-temperature char plant would be a profitable venture.

The figures given herein would change considerably for char plants in the Chicago, Pittsburgh, or other districts, but it is interesting to note that fundamentally there seems to be a reason for the use of low-temperature char for this purpose at locations where the cost of low-volatile coal is appreciably higher than the cost of a suitable char.

DISCUSSION

(*H. H. Lowry presiding*)

C. E. Lesher,[*] Pittsburgh, Pa.—I have just one question. Did you experience any difficulty in mixing the char with the coal in preparing it for charging into the high-temperature ovens?

G. V. Woody (author's reply).—As far as I know, there was no difficulty in that respect.

C. E. Lesher.—We discussed much the same subject at Cleveland last October, and we realize that this is a very, very important contribution, and to the extent that the results at Pueblo can be duplicated elsewhere it will be an important matter for all coke-oven operators making metallurgical coke.

Many of us are still wondering why it is that when a low-temperature coke is mixed with a high-volatile coking coal a good metallurgical coke is made, but when a high-temperature coke breeze is mixed with the same coal the result is quite different. I think that there is a subject here for a considerable amount of additional research.

C. C. Russell,[†] Kearny, N. J.—It was my good fortune to visit the Colorado Fuel and Iron Co. periodically to observe at first hand the work that Mr. Price and Mr. Woody have so ably described. The work covered by the report extended over a period of several years and the many details of the problem that were all thoroughly investigated with great care have been condensed into this excellent report.

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[*] President, Pittsburgh Coal Carbonization Company.

[†] Research Department, Koppers Company.

A very material improvement in the physical properties of coke is shown to be obtained by low-temperature char mixed with the high-volatile Colorado coal. One of the interesting findings of this work is that there is a critical volatile matter content of the char that has a value between 18.5 and 19.5 per cent. The fact that a satisfactory char could be made from a noncoking coal as well as from the coal regularly used for coking is an item of particular interest also.

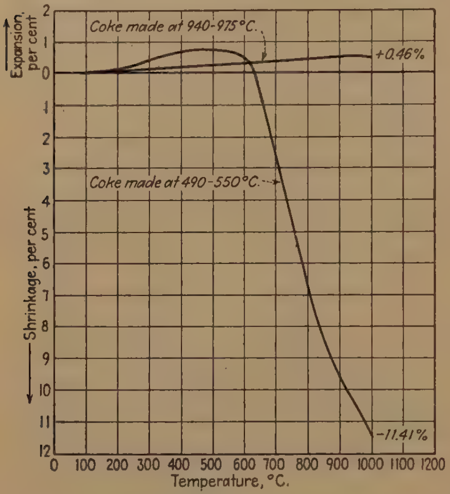

FIG. 13.—SHRINKAGE OF COKE MADE AT DIFFERENT TEMPERATURES.

The mechanics of the reactions by which the char causes coke improvement needs some clarification, for, aside from the volatile matter content, the physical and chemical properties of the char are quite different from those of low-volatile coal. Although the char may have some slight residual coking properties, it certainly does not have the swelling properties of low-volatile coal, nor would it create pressure against oven walls or expand if coked alone in an oven. It is known that char will shrink if reheated and brought up to the temperature of high-temperature coke ovens. Many high-volatile coals when coked alone produce coke that is excessively fractured, and this is due to the high over-all shrinkage of the charge. Admixture of low-volatile coal with high-volatile coal reduces the amount of fractures, owing undoubtedly to the reduction in the amount of shrinkage. It is also well known that admixture of coke breeze, fine anthracite, petroleum coke, and char with high-volatile coals reduces the amount of fractures, and here again it may be presumed that the cause is reduction in shrinkage. It is shown that coke breeze and anthracite produce coke of inferior physical properties, whereas a small improvement is obtained with petroleum coke, and with char. The question may now be raised as to why this is so.

The data in Fig. 13, obtained in our Laboratory in the study of the shrinkage of coke, shows the change in linear dimension of two pieces of coke during heating from room temperature to 1000°C. (1832°F.). The coke made in the temperature range of 940° to 975°C. shows a small but continuous expansion throughout the heating, whereas the coke made in the temperature range of 490° to 550°C. expands up to about 500°C. then starts contraction, and at 600°C. a very rapid contraction begins. Coke breeze will have the characteristics of the high-temperature coke whereas char will have the characteristics of the low-temperature coke. Prof. W. Swietoslawski has suggested, in his monograph on Coke Formation Process and Physico-Chemical Properties of Coals, that char must have the contraction characteristics of the semicoke just emerging from the plastic state, in order to produce the most satisfactory physical characteristics in the coke. The data presented appear to substantiate that idea and also indicate why coke breeze is not a satisfactory material to mix with coal to improve the physical properties of the coke.

Although Professor Swietoslawski also makes the statement that char must be made from the coal with which it is to be mixed, if the above explanation is correct it should be possible to make a satisfactory char from any coal providing that suitable heat-treatment can be found so that the char produced has substantially the same contraction chracteristics as the semicoke. This may explain why the Kebler coal produced a satisfactory char and the fact that the best results were obtained from char made from 100 per cent Kebler coal may be due to more uniform heat-treatment because of the noncoking nature of the Kebler coal.

If the contraction characteristic of the char is the essential factor in the production of satisfactory coke from coal-char mixtures, and there is a critical volatile matter of the char, the volatile matter of the char may be simply a measure of the critical contraction characteristics. It is probable that char made from other coals would have a different critical volatile matter because its critical contraction would occur at some other volatile content. The fact that the char made from the subbituminous coal tried did not give satisfactory results may have been because critical contraction occurs at some other volatile content than 19 per cent.

Another factor should not be overlooked in these considerations—the inert or infusible material mixed with the coal should have the characteristic of being readily wet by the coal when it is in the plastic state. Coke breeze may not be wetted by the plastic coal, whereas the char may be easily wetted. There is no technical information on this factor but it should not be forgotten.

The application of this process to coke-oven plants in the East will depend undoubtedly on the economics of manufacture of char as compared with the cost of low-volatile coal. The fact that great difficulty has been encountered in the production of char from strongly coking coals is a disadvantage in the East for, so far as is known, there are no noncoking coals in the Appalachian region.

F. Puening,* Bethlehem, Pa.—I think I should mention an English book by Mott and Wheeler, which was published about three years ago and of which very few copies are in this country. I happen to have one, and there are important chapters in that book on just this subject.

The results do not agree with what we have heard. Their finding is that a char made at a low temperature is really not the best, but that a medium temperature coke is by far the best substance. They undercoked coal in a high-temperature oven by 4 hr. Normal coking time was 12 hr. The oven was only 12 in. wide. They undercoked it by 4 hr. and made a medium temperature coke, which by far excelled all the other substances that could be added to coal. To wit, it was better than high-temperature coke breeze, petroleum coke, low-temperature coke of any description and low-temperature coke that had been superheated to imitate high-temperature coke breeze.

They wound up with the conclusion that high-temperature breeze is second best. They point out that high-temperature breeze is really a medium temperature coke breeze because usually it is generated from the center of the charge. The inner portions of a piece of coke are more friable, more porous, and contribute more to the quantity of the coke breeze, and these inner portions can be assumed to have been made at medium coking temperatures, and not at high coking temperatures.

However, high-temperature breeze was only second best. The very best was medium temperature coke, and thinking about this, it occurred to me that perhaps the reported observation that high-temperature breeze is not so good has not much to do with the exact coking temperature, but with the fact that the ordinary high-temperature coke breeze is very much higher in ash than a char that has been made from 100 per cent of coal.

If, for instance, I take a char made at any temperature from a 100 per cent coal, it may have 10 per cent ash. If, however, I first put the same coal into a coke oven and then collect the coke breeze, I find that it has 20 or 22 per cent ash, and we all know that coal ought to be washed and that coke is improved if ash is reduced; therefore, why should not the char made from 100 per cent coal be better than breeze collected from the remnants of high-temperature coke having at least twice as much ash?

G. V. Woody.—I want to say one thing about Dr. Puening's remarks about high-ash coke breeze. I happen to know of some experimental work that was done where the coke breeze was made from low-ash coke, and while it made better coke than high-ash breeze the results did not compare favorably with the results that have been obtained from using the low-temperature char, or medium temperature char, as the case may be.

A. C. Fieldner,* Washington, D. C.—I think we are all very fortunate that this paper has been presented and gets into the records, because it adds some experimental information that has been conducted with our American coals. Undoubtedly, as we study them we find differences between these results and others, and as we study it more we will probably find the reasons for them. It certainly is important for the future that we know much more about possibilities of blending, especially in the regions where no low-volatile coals are available.

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* Bethlehem Steel Company.

* Chief, Fuels and Explosives Service, U. S. Bureau of Mines.

MEMBER.—May I ask the size of the particles that you blend with the high-temperature coke from the Hayes ovens?

G. V. WOODY.—The Hayes oven product was about minus ½ in. as discharged. It was ground down so that about 75 or 80 per cent passed a ⅛-in. screen and then blended with the coal, which was about the same size.

Bureau of Mines Research on the Hydrogenation and Liquefaction of Coal and Lignite

By Arno C. Fieldner,* Member A.I.M.E., Henry H. Storch† and Lester L. Hirst‡

(New York Meeting, February 1944)

Experimental work on liquefaction of coal was taken up by the Bureau of Mines in 1936 when it became evident that a prudent policy from the national point of view should include preparation for the time when gradual exhaustion of petroleum resources would require supplementing the growing demand for motor fuel with gasoline and diesel oil from coal. At that time (1936) certain foreign nations that had no home sources of petroleum, especially Germany, England, and Japan, were conducting active research and at least two of them were producing liquid fuel from coal on a commercial scale. Germany was reported to have an annual producing capacity of 800,000 metric tons of gasoline per year, and England had just put into operation the Billingham plant of Imperial Chemical Industries for the hydrogenation of bituminous coal and tar by the Bergius-I.G. process. The annual capacity of this plant was about 150,000 long tons of gasoline per annum (3500 bbl. per day).

With these developments taking place in foreign countries, it seemed important to undertake a long-range program of fundamental research in this country to establish a sound groundwork for the evaluation of foreign work on synthetic liquid fuels and at the same time to obtain data on the oil yields and operating difficulties using various kinds of American coals and lignite to the hydrogenation process. The results of such research would be useful in determining the yields and quality of liquid products obtainable from various coals so that American industry would be in position to select the most suitable coals and make rapid progress in establishing this new industry when the commercial need should arise. It was hoped, also, that progress in a better understanding of the chemistry of coal hydrogenation would reduce the cost of the process materially.

To carry out this program, a small continuous unit capable of hydrogenating 100 lb. of coal in 24 hr. was installed at the Central Experiment Station of the Bureau of Mines at Pittsburgh, Pa. Small steel autoclaves or bombs of 0.3 gal. (1.2 liters) capacity also were used for batch-hydrogenation experiments in the study of the chemical fundamentals of the process. Much valuable assistance was obtained in planning the experimental program and designing the equipment from the Fuel Research Station of Great Britain and the Fuel Laboratories of the Mines Department of Canada, where such investigations had been in progress for several years.

Equipment and General Procedure of Research

The industrial liquefaction of coal by the Bergius-I.G. hydrogenation process as practiced at Billingham, England, from

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1935 to 1937, is carried out in two stages; first, in the primary or liquid phase, and, second, in the vapor phase. Liquid-phase hydrogenation is accomplished by mixing a small amount—usually less than 0.1 per cent of the weight of the coal—of a suitable catalyst, preferably a compound of tin, with approximately equal parts of powdered coal and heavy oil previously obtained in the process, and pumping this paste through a preheater in a high-pressure steel vessel through which hydrogen gas also is pumped at a pressure of about 3000 lb. per sq. in. The preheater raises the temperature of the reacting mass to about 860°F. (460°C.). The oil product from the reaction vessel is separated from the solid residue of ash-forming material and unliquefiable carbonaceous matter, and is distilled into light, middle, and heavy-oil fractions.

The light oil is refined to gasoline. The middle oil is hydrogenated further by pumping it in the vapor phase together with hydrogen at a similar high pressure and temperature through a second high-pressure vessel containing a fixed catalyst, such as molybdenum or tungsten sulphide. The heavy oil goes back into the process to serve as the liquid vehicle that is mixed with the powdered coal.

The octane rating of the gasoline obtained in the process as operated a few years ago was 70 to 75. By the addition of 3 c.c. of tetraethyl lead per gallon the octane number can be increased to 80 or 85. Iso-octane synthesized from the butanes produced as a by-product in the process may be added to the gasoline along with tetraethyl lead. The resultant product is a very high-grade aviation fuel.

Four to five tons of high-volatile bituminous coal, 7 tons of subbituminous coal, or 9 tons of lignite are required to produce one short ton of gasoline (0.6 to 0.8 short ton of bituminous coal, 1 ton of subbituminous coal, or 1.3 ton of lignite per barrel of gasoline). These figures include the coal needed for power, steam, and hydrogen used in the process. These requirements are based on the assumption that oils heavier than gasoline will be recycled until all are hydrogenated to form gasoline and lighter products. The vapor-phase step may be omitted and the middle oil from the liquid-phase stage may be separated by distillation into diesel-oil and fuel-oil fractions.

Since the primary objective of the Bureau of Mines research was the assay of the hydrogenating properties of the various American coals and lignites, most of the experiments were confined to the primary or liquid-phase stage of hydrogenation. In other words, the liquefaction of coal was conducted in most instances to produce a crude oil only. A small amount of vapor-phase hydrogenation was carried out during the last year, and some of this gasoline was used with satisfactory results in automobiles, tanks, and airplanes.

The hydrogenation and liquefaction of coal is a complex chemical process. To produce a petroleum-like material, the ratio of the number of hydrogen atoms to carbon atoms in bituminous coal (which is about 0.83 for bituminous coal and 1.66 for petroleum) must be doubled. Also, most of the oxygen, nitrogen, and sulphur must be removed, and the coal molecules must be dissociated or "cracked" until their molecular weight approximately equals that of petroleum molecules. Prior to actual thermal "cracking," depolymerization and colloidal dispersion of the coal substance no doubt takes place, owing to the solvent action of the liquid vehicle, which also may act as a carrier of hydrogen in the converter. The dissolved hydrogen or a component of the vehicle which readily yields hydrogen reacts immediately with the coal molecules while they are in the dispersed state and thus prevents repolymerization and formation of solid aggregates. Many of the chemical reactions are speeded up by a suitable catalyst and different catalysts effect different reactions, thus influencing the composition of the products. It is obvious that a scientific study of the fundamentals of coal hydrogenation should include experiments on the effect of numerous variables, such as temperature, pressure, time of reaction, nature of vehicle, and composition of catalysts as well as the petrographic composition of the coal. Much of this research is best conducted with small rotary autoclaves, in which conditions can be controlled accurately. The results obtained are indicative of those in a commercial plant but must be supplemented by continuous hydrogenation in a laboratory-scale pilot plant designed to follow the flowsheet proposed for commercial operation. Both methods were followed in the Bureau program.

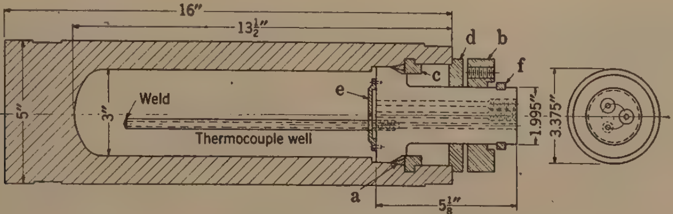

FIG. 1.—CROSS SECTION OF HYDROGENATION BOMB.

BATCH HYDROGENATION IN SMALL ROTATING AUTOCLAVES

Description of Apparatus and General Experimental Procedure

The autoclave used in the Bureau experiments consisted of a cylindrical bomb of 0.3 gal. (1.2 liters) capacity forged from 18 Cr, 8 Ni stainless steel. Its construction is shown in Fig. 1 and the rotating mechanism and electric heater appear in Fig. 2.

The following procedure was used in most of the experiments.[1] A mixture of 1.75 to 3.50 oz. (50 to 100 grams) of pulverized coal, an equal amount of tetrahydronaphthalene, and 1 per cent by weight of the coal of stannous sulphide was placed in the autoclave. After the air had been flushed from the autoclave with hydrogen, more hydrogen was introduced until the desired initial pressure of 1000, 1500 or 1800 lb. per sq. in. at 68°F. (20°C.) was reached. The hydrogen line was then disconnected and the rotating autoclave was heated to the desired maximum temperature at the rate of 3.6° to 5.6°F. (2° to 4°C.) per minute, and maintained there for 3 hr. The maximum temperature in most of the experiments was 725° to 806°F. (385° to 430°C.). After the bomb had cooled to room temperature, the gaseous, liquid and solid contents were separated, measured and analyzed. The percentage liquefaction is defined as 100 minus the percentage of acetone and benzol-insoluble material. This residue consisted of the ash-forming material plus the unliquefiable carbonaceous matter. The oil was distilled up to 419° or 428°F. (215° or 220°C.). The distillates were mixtures of tetrahydronaphthalene vehicle, small amounts of naphthalene, and light oils from the coal, and the distillation residues were black, semisolid pitches. The distillates were analyzed for tar acids, tar bases, olefins, and aromatics.

The most significant result obtained in the autoclave tests is the *percentage liquefaction,* which includes all gaseous and liquid products and such solid products as are soluble in acetone or benzol.

Relation of Rank and Type of Coal to Yield and Composition of Liquefaction Products

It has been generally recognized by European and British investigators that the degree of liquefaction of a coal in the hydrogenation process varies inversely with the rank of the coal; that is, with the extent of metamorphism from lignite to anthracite. Less has been published on the effect of the type of the coal as expressed by its petrographic composition and the nature of the plant constituents from which it originated. The late Dr. Reinhardt Thiessen, outstanding authority on the paleobotanical composition of coal, suggested that variations in the plant entities or petrographic constituents of coals in many instances probably would obscure the effect of rank in the examination of coals in the progressive series from lignite to anthracite. Therefore, in the Bureau research attention was given both to the rank and the type of the coals, and liquefaction tests were made of carefully selected samples of the banded and petrographic constituents of coals of various ranks.

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[1] References are at the end of the paper.

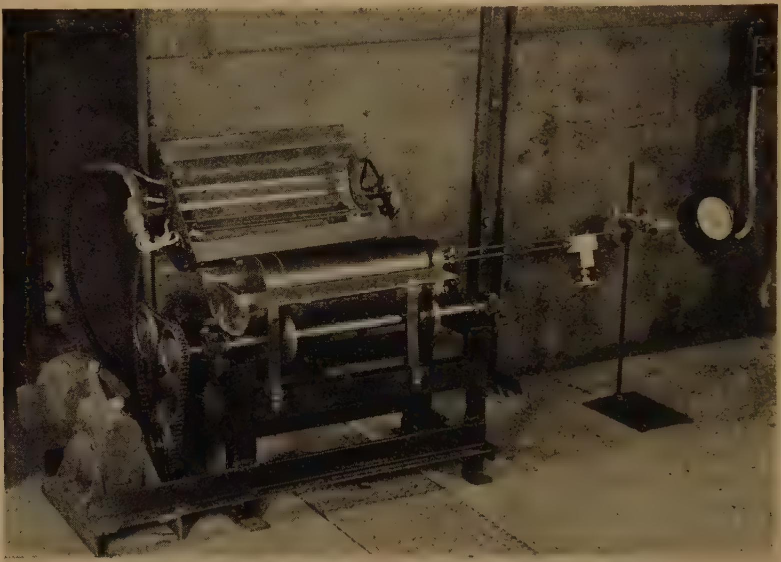

FIG. 2.—HYDROGENATION BOMB AND ACCESSORY EQUIPMENT.

Effect of Rank on the Hydrogenation of Anthraxylon and Bright Coals

Anthraxylon (vitrain), the principal constituent of bright coals, and bright coals were selected for the study of the effect of rank on the hydrogenation of coal. This procedure minimized the obscuring effect of differences in petrographi composition. The coals ranged in rank from peat to anthracite (carbon content from 55.5 to 94.9 per cent of the dry, ash-free coal). From these investigations, Storch[1,2] and his associates have drawn the following conclusions on the effect of rank as indicated by the total carbon content of the dry, ash-free coal on various features of the liquefaction of coal by hydrogenation:

1. *Degree of Liquefaction.*—Peat, lignite, subbituminous coal and high-volatile bituminous coal are almost completely "liquefied." Fig. 3 shows that the percentage liquefaction at optimum temperature and pressure varies from 95 to 98 per cent for anthraxylons and bright coals in the range of rank below 90 per cent carbon. Owing to the advanced degree of coalification, decreased chemical activity, and increased thermal stability, coals containing more than 90 per cent carbon were less com-

pletely liquefied, as shown by the sharp upswing of the curve in Fig. 3.

2. *Yield of Oil and Pitch.*—On account of the difficulty of making accurate separation of the light oil distilling up to 428°F. (220°C.) from the tetrahydronaphthalene ($C_{10}H_{12}$) vehicle (boiling point 404°F. or 207°C.), no attempt was made to measure the yield of these light oils. From the hydrogen mass balance, it was calculated that these oils comprise 10 to 30 per cent of the yield of pitch. The black, semisolid or solid residue from this distillation, designated as pitch, comprised 30 to 80 per cent by weight of the dry, ash-free anthraxylons, and from 35 to 70 per cent of the dry, ash-free bright coals. As shown in Fig. 4, the yield of pitch is directly proportional to the carbon content up to about 90 per cent carbon. This point approximates the boundary line or rank between high-volatile and medium-volatile bituminous coal. Beyond this point the yield of pitch decreases because liquefaction is incomplete and high yields of insoluble carbonaceous residues are obtained. Coals containing more than 89 per cent carbon, on the dry, ash-free basis, such as medium-volatile and low-volatile bituminous coals and anthracitic coals are unsuitable under conditions appropriate for commercial hydrogenation.

The viscosity of the pitches increased with the rank of the coal. The pitches from high-rank anthraxylons were brittle, friable solids at room temperatures. Those from low-rank coals were semisolid. Since this heavy oil or pitchlike product is used as a vehicle, or "pasting-oil," in the commercial hydrogenation of coal, its viscosity is of practical importance.

3. *Yield of Gases and Consumption of Hydrogen.*—The total yield of gases, excluding hydrogen, decreased from a maximum of about 24 per cent by weight for peat to a minimum of about 1 per cent for anthracite. The decrease in gas yield has a general relationship to increase in rank of the coal. This decrease is largely on account of the decrease in yield of carbon dioxide as the oxygen content of the coal decreases. The removal of oxygen as carbon oxides instead of as water is a very fortunate circumstance in the hydrogenation of lignite and subbituminous coal, because it greatly reduces the excess hydrogen that otherwise would be required in the utilization of these low-rank high-oxygen coals. About the same amount of hydrogen was required per ton of dry, ash-free coal to liquefy coals of different rank. The hydrogen consumption was affected much more by the pressure than by the rank of the coal. In the continuous hydrogenation experiments to be described later, the consumption of hydrogen per

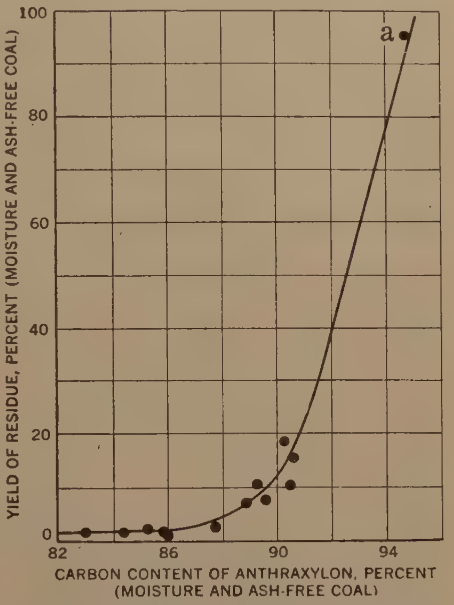

FIG. 3.—RELATION BETWEEN CARBON CONTENT OF ANTHRAXYLON AND YIELD OF INSOLUBLE RESIDUE PRODUCED BY HYDROGENATION AT 806°F. (430°C.), 1000 POUNDS PER SQUARE INCH INITIAL HYDROGEN PRESSURE.

a. Yield of residue obtained by hydrogenation of anthraxylon from anthracite, 1800 lb. per sq. in. initial hydrogen pressure, was more than 95 per cent.

ton of oil produced was somewhat greater for lignites than for bituminous coals.

4. *Yield of Phenols and Tar Acids.*—The yield of phenols in per cent by volume of the distillate up to 428°F. (220°C.) increased with decreasing carbon content, from an average of about 2 per cent for low-volatile and medium-volatile bituminous coals to about 6 per cent for lignite. It is the increasing content of oxygen as the rank of coals decreases that is responsible for the increasing content of phenols and tar acids in the hydrogenation products. These compounds can be obtained in amounts large enough to be of commercial importance as raw material for the plastics industry.

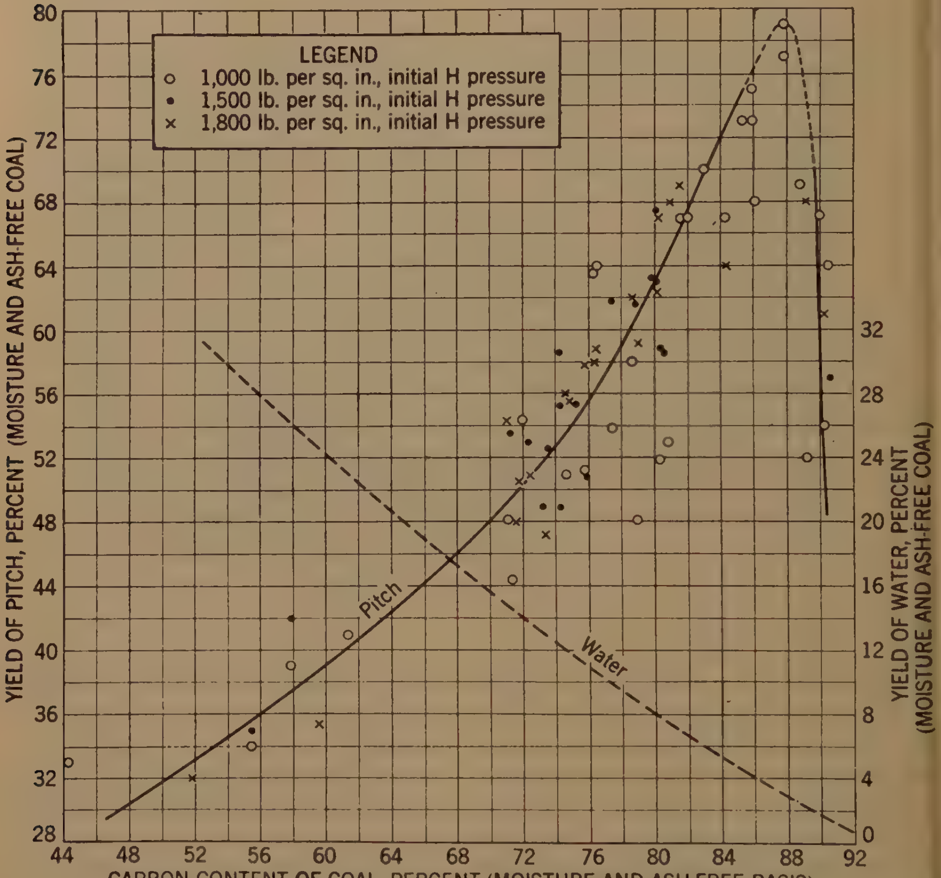

FIG. 4.—RELATION BETWEEN CARBON CONTENT OF COAL AND YIELD OF PITCH AND WATER PRODUCED BY HYDROGENATION.

Effect of Type and Petrographic Constituents on Hydrogenation of Coal

The experiments with anthraxylon and bright coals showed that they were almost completely liquefied under appropriate conditions in the range of rank from lignite to and including high-volatile bituminous coal. As the fixed carbon or total carbon content increases above this rank, as in medium-volatile, low-volatile, and anthracitic rank, the anthraxylon becomes progressively more refractory to liquefaction. Anthraxylon is the principal petrographic constituent of bright coal, which comprises most of our American common banded coals. About 50 to 75 per cent of the petrographic composition of bright coals is anthraxylon. Most of the remainder is translucent attritus, which also is liquefied to a high degree in the ranks below medium-volatile bituminous coal. The content of opaque attritus plus fusain seldom exceeds 20 per cent.

Splint and semisplint coals, on the other hand, contain much less anthraxylon—usually less than 50 per cent—and much larger proportions of opaque attritus. Banded coals containing 30 per cent or more of opaque attritus are classified as splint coals. Coals with a content of opaque attritus between that of bright coal (under 20 per cent) and splint coal (over 30 per cent) are designated as semisplint. Pure opaque attritus, the characteristic constituent of splint coals, is very difficult to separate from the coal, hence its chemical and hydrogenation properties were not determined by a study of pure samples. Therefore the properties of opaque attritus were determined by hydrogenating splint coals that contained 60 to 80 per cent of this constituent. Many splint coals contain appreciable quantities of translucent degradation matter, resins, spore-coats, and oil algae and experiments described later show that these substances are liquefied readily with high yields of oil.

Influence of Content of Opaque Attritus on the Degree of Liquefaction

The results of liquefaction tests on a series of splint and semisplint coals indicated relatively low liquefaction yields. The amounts ranged from 60 to 87 per cent of the dry-ash-free coal as compared with 95 to 98 per cent for anthraxylons from bright coals of high-volatile bituminous and lower rank. The average liquefaction of opaque attritus from bituminous coals (at 806°F. for 3 hr. in the presence of stannous sulphide, and with an initial hydrogen pressure of 1000 lb. per sq. in.) was about 60 per cent; however, the liquefaction yield for individual samples of opaque attritus ranged from about 40 to 80 per cent. It is evident from this work and that of other investigators that splint coals are not as suitable for hydrogenation as bright coals.

Hydrogenation of Hand-picked Fusains.—Of all the petrographic constituents of coal, fusain has the highest degree of opacity to transmitted light in thin slides under the microscope, the least chemical reactivity, the lowest content of hydrogen and oxygen, and the highest carbon content in coals of the same rank.

In order to determine the degree of liquefaction of fusain seven different samples were isolated from bituminous coals from the eastern states and one from a Wyoming subbituminous coal. These specimens, tested by the usual procedure of batch hydrogenation in the small autoclave showed from 10 to 55 per cent liquefaction. The highest liquefaction was obtained with the lowest rank of coal tested, a subbituminous C coal from Wyoming, and the lowest liquefaction was obtained with a fusain taken from a medium-volatile bituminous coal from the Beckley bed of West Virginia. It appears that increasing carbon content and increasing rank of the parent coal bed leads to increasing carbon content of the fusain layers in it, and a roughly parallel reduction in their degree of liquefaction. It is evident also that the presence of translucent matter in the fusain contributes to increased hydrogenation yields. Microscopic examination of residues from the hydrogenation of fusain indicated that practically all of any translucent matter visible in the fusain before hydrogenation had disappeared.

Hydrogenation of Cannel Coals and Their Petrographic Constituents

Autoclave tests of eight different samples of cannel coals indicated that 68 to 93 per cent of the dry, ash-free organic matter was liquefied. The average percentage was 84. These results are somewhat less—from 5 to 8 per cent in most instances—than for bright coals. The lower yield probably is due to the frequency of larger percentages of opaque attritus being found in many cannel coals. The opaque attritus in the samples tested ranged from 4 to 49 per cent. Usually, the predominant petrographic constituent of cannel coals was translucent attritus that contained large quantities of spore-exines, translucent degradation matter or resins. Hydrogenation tests of spore-exines separated from coals, and modern lycopodium spores indicated that these constituents were liquefied to the extent of about 96 per cent. Resins from coal and from modern pine trees were almost completely liquefied; that is, 98 to 100 per cent.

No typical boghead coals were available for test. A boghead cannel from West Virginia gave only 77.2 per cent liquefaction yield, but this sample contained 30 per cent of opaque attritus, which would detract materially from its amenability to liquefaction. An oil-shale sample from Estonia, which contained a large percentage of oil algae and therefore might be considered to be a boghead coal, was easily liquefied, the yield being 98 per cent. Therefore, it is probable that oil-algae remains in coal can be almost completely liquefied. The nonbanded cannel and boghead types of coal, in general, are liquefied to a lesser extent than bright coals of the same rank but to a greater extent than splint coals.

Summary of Ease of Liquefaction of Coals of Difficult Ranks and Types and Their Petrographic Constituents.—A summary of the ease of liquefaction of coals of different ranks and types and of their principal petrographic constituents is given in Table 1. The range of liquefaction yields given in the table are based on the batch hydrogenation experiments made by the Bureau of Mines and are to be considered as tentative only, as the tests did not cover enough different coals for final figures. The average yields for high-volatile bituminous coals probably are above the median figure of the range and the average yields for subbituminous coals probably are below the median. Most fusains mechanically separated from bituminous coals probably will be between 10 and 25 per cent liquefiable. In the petrographic analysis of coals, the reported fusain does not include the small amounts of translucent matter which adheres to hand-picked fusain and therefore the petrographically estimated fusain is assumed to contain no liquefiable carbonaceous matter.

TABLE 1.—*Liquefaction Yields of Coals of Different Ranks and Types and of Their Petrographic Constituents*

RANK, TYPE, OR PETROGRAPHIC CONSTITUENT	LIQUEFACTION YIELD, PER CENT BY WEIGHT OF DRY, ASH-FREE COAL, USUAL RANGE
Anthracitic rank	Less than 70
Bituminous low-volatile rank	70 to 90
Bituminous medium-volatile rank	75 to 95
Bituminous high-volatile and subbituminous ranks	85 to 96
Lignitic rank	90 to 97
Bright coal type below medium-volatile rank	85 to 97
Splint and semisplint coal types	65 to 85
Cannel and boghead coal types	70 to 98
Translucent anthraxylon, translucent attritus, spore-exines, and algae remains	95 to 98
Resins	98 to 100
Opaque attritus	40 to 80
Fusain	10 to 50

From this investigation it is evident that only the common banded bright coals ranging in rank from lignitic to high-volatile bituminous inclusive are to be considered as potentially suitable for commercial hydrogenation. A few cannel and boghead coals that are low in fusain and opaque attritus may be suitable and, of course, many bright coals may be unsuitable because of a high content of fusain and ash-forming material. Splint coals and all types of coal above high-volatile bituminous in rank are not suitable for liquefaction by direct hydrogenation although they can be used for the production of synthetic liquid fuels by converting them to water gas and then hydrogenating the carbon monoxide.

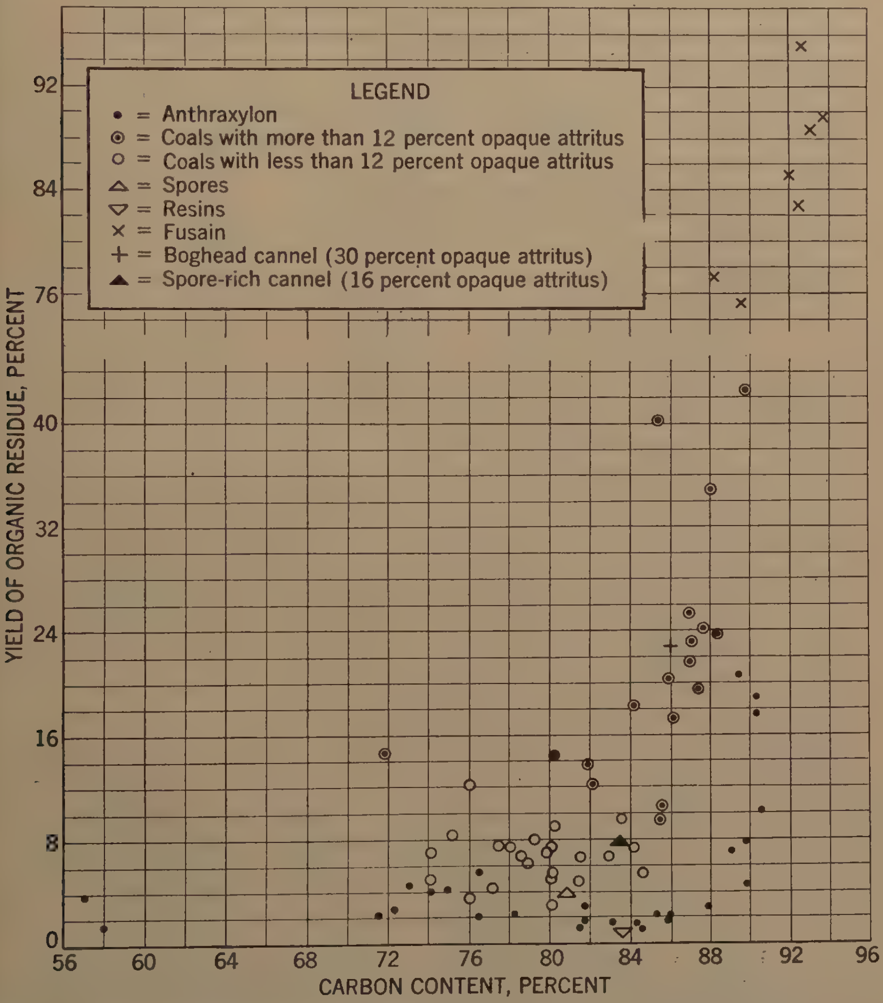

FIG. 5.—RELATION BETWEEN CARBON CONTENT OF PETROGRAPHIC CONSTITUENTS AND TYPES OF COAL, AND YIELD OF INSOLUBLE ORGANIC RESIDUE ON DRY, ASH-FREE BASIS.

Approximate estimates may be made of the degree of liquefaction from the total carbon content of the dry, ash-free coal sample and from the petrographic composition, as shown in Figs. 5 and 6. These graphs are based on the results of the Bureau of Mines tests. Fig. 5 gives the relation between the yield of insoluble organic residue and the carbon content of the dry, ash-free coal. Fig. 6 gives the relation between the yield of insoluble residue (organic plus inorganic) obtained in the hydrogenation tests and that calculated from the sum of the ash plus fusain plus

38 per cent of the opaque attritus. While the results thus obtained are only rough estimates, they are adequate for rejecting highly unsuitable coals.

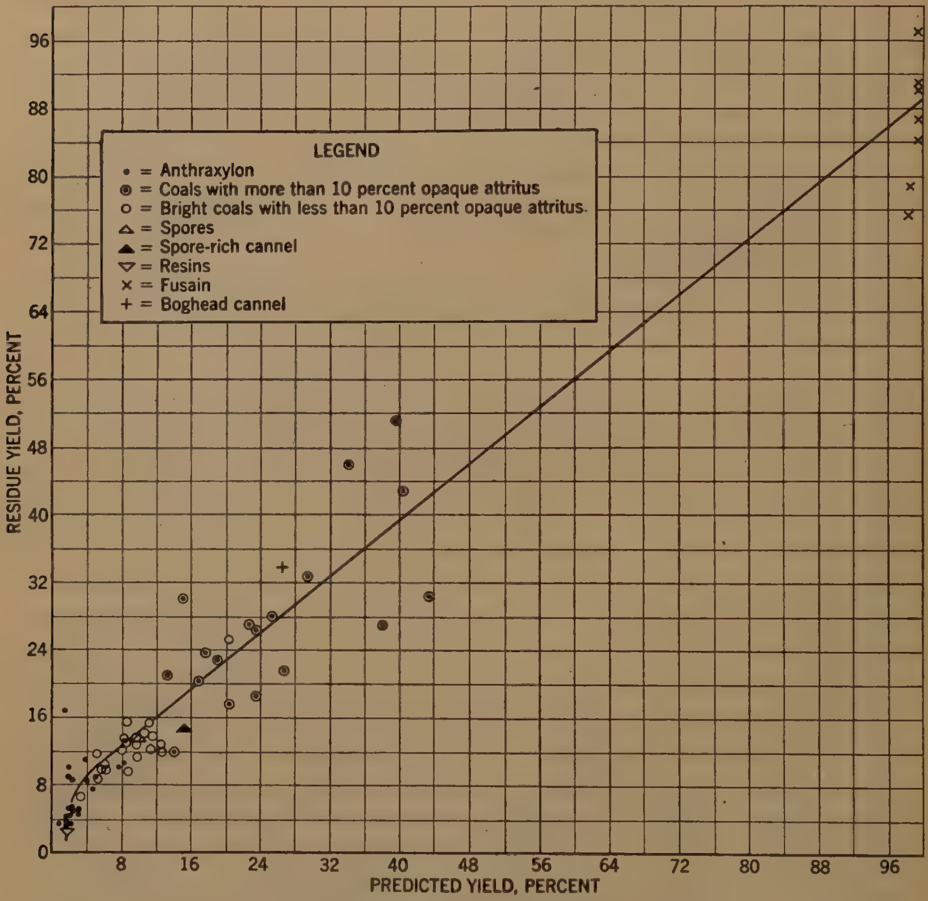

Fig. 6.—Yields of residue predicted from petrography.

Continuous Hydrogenation in a 3-inch Converter Experimental Unit[3]

Apparatus and Method

The experimental unit constructed at the Pittsburgh Station of the Bureau of Mines was patterned after a similar unit designed by the Fuel Research Station of Great Britain. It has a daily capacity corresponding to the liquid-phase hydrogenation of about 100 lb. of coal from which, with further vapor-phase hydrogenation, about 8 gal. of gasoline can be obtained.

To liquefy coal at industrially feasible rates, contact of coal, catalyst, liquid vehicle, and hydrogen at a temperature of 752° to 842°F. (400° to 450°C.) and a pressure of 3000 to 5000 lb. per sq. in. are necessary. A flow diagram of the coal-liquefaction unit is shown in Fig. 7. The coal sample is crushed, dried, mixed with catalyst, pulverized in a ball mill to 200 mesh, and mixed with a little more than an equal weight of the liquid vehicle (heavy oil from previous runs). The resultant paste is pumped continuously at a temperature of 212°F. (100°C.) into the bottom of the converter with a high-pressure paste pump.

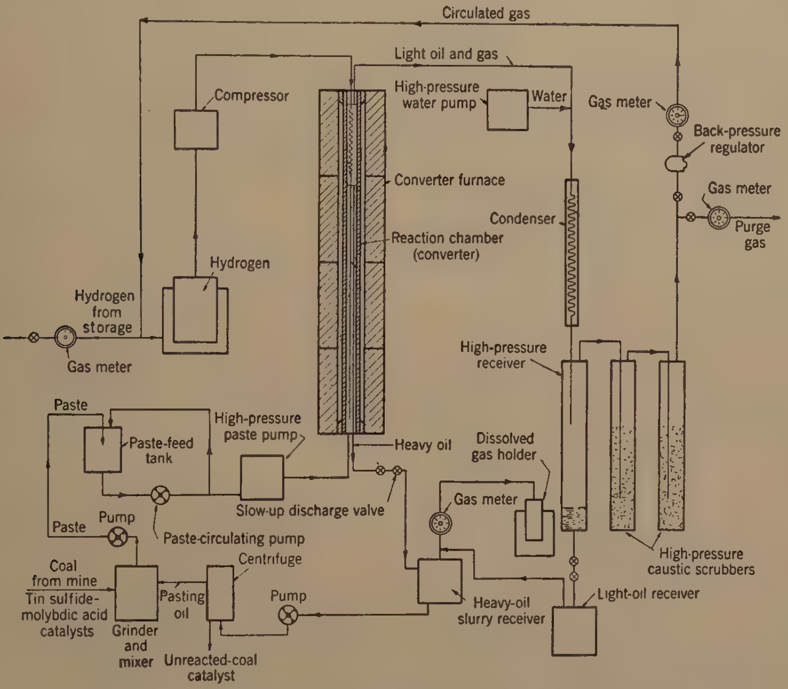

FIG. 7.—SCHEMATIC FLOW DIAGRAM, BUREAU OF MINES EXPERIMENTAL COAL HYDROGENATION UNIT, PITTSBURGH.

The converter is a forged-steel tube made of 18 per cent Cr, 8 per cent Ni, low-carbon alloy steel. The length is 8 ft., the bore 3 in., and the wall thickness 1 in. It is heated electrically by a resistance-wound furnace.

The hydrogen is introduced through a pipe that enters the top of the converter, passes through a preheating coil and then continues in a straight pipe to near the bottom, where it is discharged into the coal suspension and serves to agitate the charge on its passage to the top. On leaving the converter, the gases and light oil vapors pass through a waterjacketed condenser to the high-pressure receiver. The light oil collected in this receiver is termed "overhead oil," to distinguish it from the less volatile and heavier oil that is discharged as a heavy-oil slurry containing the catalyst and ash-forming material, and carbonaceous residue from the coal.

To reduce its viscosity, the discharged slurry is diluted with some (about 30 per cent of the volume of the slurry) of the middle oil obtained by fractional distillation of the converter overhead oils, and is then centrifuged. The centrifuged oil is used to provide the vehicle for producing the mixture of coal and oil at the head of the process.

The oil vapors carried through the top of the converter by the unreacted hydrogen gas consist of about 20 per cent gasoline boiling below 492°F. (200°C.) and 80 per cent of middle oil boiling up to 626°F. (330°C.). This overhead oil plus hydrogen is passed through a water-cooled condenser and then into the high-pressure receiver where separation takes place.

Liquid-phase Hydrogenation of Pittsburgh-bed Coal

Because Pittsburgh-bed coal is widely distributed in Pennsylvania, Ohio, West Virginia, and Maryland, and because of its similarity in rank and type to the Durham bituminous coal used in the Billingham commercial hydrogenation plant in England, it was chosen for the first coal to be used in working out a procedure suitable for the assay of American coals for hydrogenation. The samples were secured from the Bureau's Experimental Mine at Bruceton, Pennsylvania.

A large number of runs under various operating conditions showed the following results:

1. Increasing the temperature from 788° to 858°F. (420° to 459°C.) decreased the oil yield (benzene-soluble material) from 71 to 53 per cent and increased the gas yield from 10 to 37 per cent of the dry, ash-free coal. However, decrease in oil yield was more than compensated by the increase in gas yield. Liquefaction in the sense of decomposition of the coal increased with temperature, the percentage of ash-free benzene-insoluble residue decreasing from 16.3 to 6.3 per cent of the dry, ash-free coal.

2. Increase of contact time produced a marked increase of gas yield from 15 per cent at 1.6 hr. contact time to 30 per cent at 4 hr. contact time and a decrease of oil yield from 73 per cent at 1.6 hr. contact time to 63 per cent at 4 hr. contact time. For contact times less than 1.6 hr. at 824°F. (440°C.), this trend appears to reverse itself, so that shorter contact times lead to lower oil yields. The effects of temperature and contact time are analogous, a higher temperature being equivalent to a longer contact time. From a practical point of view the limiting low temperature is determined by the time of contact desired and the viscosity of the heavy-oil slurry. The slurry obtained at 788°F. (420°C.) was as viscous as could be discharged conveniently at 212°F. (100°C.). Greater cracking of hydrocarbons at the higher temperatures is evident in the increase of the overhead to heavy-oil ratio from 0.41 at 788°F. (420°C.). to 1.72 at 858°F. (459°C.). Similar effects are produced by increase of contact time.

In the operation of the experimental unit for the liquefaction assay of coals, the overhead to heavy-oil ratio was so adjusted that the amount of the latter equaled the weight of vehicle mixed with the coal charged into the converter.

Reviewing all the continuous hydrogenation tests made with the Pittsburgh-bed coal from Bruceton, Pa., it is concluded that the optimum conditions for liquefaction at 824°F. (440°C.) and 200 to 300 atmospheres pressure are approximately 2.00 hr. contact time and circulation of about 100 cu. ft. (measured at 68°F. and 30 in. mercury pressure) per hour of hydrogen. Under these conditions the yields of products given in Table 2 were obtained in a single pass through the converter.

TABLE 2.—*Average Yield under Optimum Conditions from the Liquid-phase Hydrogenation of Pittsburgh-bed Coal from U. S. Experimental Mine*

PRODUCT	PERCENTAGE OF DRY, ASH-FREE COAL PLUS HYDROGEN ABSORBED
Oil product	64–68
Gaseous hydrocarbons, methane to butane inclusive, plus carbon oxide gases	20–25
Oxygen, nitrogen, and sulphur hydrogenated to water, ammonia, and hydrogen sulphide	3–6
Unreacted fusain and opaque attritus	6–8
COMPOSITION OF OIL PRODUCT	PERCENTAGE BY VOLUME
Oil boiling in the gasoline range below 401°F. (205°C.)	20
Oil boiling from 401° to 626°F. (205° to 330°C.)	70
Tar acids	15–18
Tar bases	3–5
Neutral oil	77–82
COMPOSITION OF NEUTRAL OIL	PERCENTAGE BY VOLUME
Olefins	3–5
Aromatics	67–70
Saturated hydrocarbons (paraffins plus naphthenes)	22–27

In general, no exact specifications can be given as to the optimum temperature and contact time for a given coal. There is a fairly wide range of temperatures at which the contact time may be adjusted to give the same yields and the same viscosity of heavy-oil slurry. This temperature range for the Experimental Mine coal is approximately 788° to 824°F. (420° to 440°C.); the corresponding contact-time range is 2.75 to 1.75 hours.

Liquid-phase Hydrogenation Assays of High-volatile Bituminous, Subbituminous, and Lignitic Coals

After developing a suitable standard procedure for the liquid-phase hydrogenation assay of coals, the next step was to apply this method to a survey of typical American coals ranging in rank from high-volatile A bituminous coal to lignite.[4] These are described in Table 3. Special attention was given to subbituminous coals and lignites because large reserves of these ranks of coal do not occur in Germany or Great Britain, and for this reason they have not been used in the British or German liquefaction plants. This country has very large reserves of these lower rank coals and they are not suitable for the manufacture of coke or gas. Neither do they lend themselves to transportation over long distances or to storage, because of their weathering and slacking properties. Their utilization for the production of liquid fuels would seem to be an ideal use for these coals when such conversion becomes economically possible.

YIELDS OF PRODUCTS

The yields obtained under the optimum conditions of hydrogenation for each coal are given in Table 4. The coals are arranged in order of the total carbon content of the dry, ash-free coal, the highest carbon content being at the top of the table and the lowest at the bottom. This arrangement places the coals in descending order of rank. The yield of oil based on percentage of dry, ash-free coal (column 12) showed, in general, decrease with decrease in rank below that of the West Virginia No. 21 mine coal. The lowest yield in coals of high-volatile bituminous rank was 62 per cent for the coal from Sayreton, Ala. and the highest yield was 74 per cent for that from Columbia, Utah. The other six high-volatile bituminous coals gave oil yields between 69 and 71 per cent, inclusive. The Sayreton coal would be expected to give the lowest yield of the high-volatile group because it is of distinctly higher rank than the others, verging on medium-volatile rank; likewise, the Columbia, Utah, coal would be expected to give the highest oil yield because of its relatively low position in the high-volatile rank and its high content of resin, which hydrogenates readily.

On the same moisture-and-ash-free basis, the three subbituminous coals gave lower oil yields, ranging from 63 to 67 per cent, and the two Dakota lignites gave the lowest yields, 54 and 55 per cent.

However, in actual practice the oil yields based on coal as mined or washed will be progressively less than the results just given, on account of the increasing moisture content of the low-rank coals. Lignites contain from 35 to 45 per cent moisture.

Column 16 of Table 4 gives the oil yields in gallons per ton of coal as mined. This is the crude oil from the assay tests. The total average yield of gasoline obtainable from this crude, including vapor-phase hydrogenation of the middle oils, is estimated at 70 per cent of the oil yields given in columns 12 and 16. None of these yields include the coal or other fuel required for the generation of additional hydrogen, steam, and power. Usually, this additional coal is estimated as equal to the coal liquefied, so that the net yields of liquid fuel per ton of total coal used for all purposes should be taken as one half of the

TABLE 3.—*Origin and Chemical and Petrographic Analyses of Coals Tested*

State	County	Bed	Mine	Rank[a]	Condition[b]	Proximate Analysis, Per Cent				Ultimate Analysis, Per Cent					Calorific Value, B.t.u.	Petrographic Analysis, Per Cent			
						Moisture	Volatile Matter	Fixed Carbon	Ash	Hydrogen	Carbon	Nitrogen	Oxygen	Sulphur		Anthraxylon	Attritus		Fusain
																	Translucent	Opaque	
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Alabama	Jefferson	Mary Lee	Sayreton	BHVA	1	4.2	27.0	59.0	9.8	5.2	74.7	1.5	8.0	0.8	13,550	61	27	5	7
					2		28.2	61.6	10.2	4.9	77.9	1.6	4.6	0.8	14,140				
					3		31.4	68.6		5.4	86.9	1.8	5.0	0.9	15,490				
West Virginia	Monongalia	Upper Freeport	Ind. Col. No. 21	BHVA	1	3.2	29.2	59.3	8.3	5.1	76.6	1.3	7.9	0.8	13,600	67	26	6	1
					2		30.2	61.3	8.5	4.9	79.4	1.4	5.0	0.8	14,050				
					3		33.0	67.0		5.3	86.6	1.5	5.7	0.9	15,400				
Alabama	Walker	Black Creek	Empire	BHVA	1	2.7	35.4	59.1	2.8	5.4	79.8	1.7	9.5	0.8	14,310	45	38	7	10[d]
					2		36.4	60.7	2.9	5.3	82.0	1.7	7.3	0.8	14,710				
					3		37.4	62.6		5.4	84.4	1.8	7.6	0.8	15,150				
Pennsylvania	Allegheny	Pittsburgh	U. S. Experimental	BHVA	1	1.6	35.7	56.4	6.3	5.3	77.6	1.6	7.6	1.6	13,870	63	22	12	3
					2		36.3	57.3	6.4	5.2	78.8	1.6	6.4	1.6	14,090				
					3		38.8	61.2		5.6	84.2	1.7	6.8	1.7	15,050				
Utah	Carbon	Lower Sunnyside	Columbia	BHVA	1	5.5	39.2	50.4	4.9	5.7	73.5	1.4	13.5	1.0	13,010	30	61	6	3
					2		41.4	53.4	5.2	5.4	77.8	1.5	9.1	1.0	13,630				
					3		43.7	56.3		5.7	82.1	1.6	9.5	1.1	14,630				
Illinois	Franklin	No. 6	Orient No. 2	BHVB	1	7.8	33.7	52.0	6.5	5.5	69.0	1.6	15.5	1.9	12,240	70	24	1	5
					2		36.5	56.4	7.1	5.0	74.9	1.7	9.2	2.1	13,280				
					3		39.3	60.7		5.3	80.6	1.9	10.0	2.2	14,290				
Indiana	Vigo	No. 4	Saxton No. 1	BHVC	1	13.6	34.2	44.9	7.3	6.0	64.0	1.4	19.6	1.7	11,480	65	24	7	4
					2		39.5	52.0	8.5	5.3	74.0	1.6	8.6	2.0	13,280				
					3		43.2	56.8		5.7	80.9	1.7	9.5	2.2	14,510				
Washington	King	McKay	Strain-Upper Diamond	BHVC	1	9.6	40.1	46.6	3.7	6.2	68.9	1.9	18.7	0.6	12,350	65	34	1	0
					2		44.4	51.5	4.1	5.7	75.3	2.1	12.1	0.7	13,660				
					3		46.3	53.7		5.9	78.6	2.2	12.5	0.8	14,240				
Montana	Rosebud	Rosebud	Colstrip	SBB	1	25.1	28.5	38.8	7.6	6.3	51.7	0.7	33.2	0.5	8,900	51	33	13	3
					2		38.0	51.9	10.1	4.6	69.0	1.0	14.6	0.7	11,880				
					3		42.3	57.7		5.2	76.7	1.1	16.2	0.8	13,210				
Colorado	Weld		Puritan	SBB	1	24.1	29.1	42.7	4.1	6.3	55.0	1.2	33.1	0.3	9,460	31	55	10	4
					2		38.3	56.2	5.5	4.7	72.1	1.5	15.8	0.4	12,470				
					3		40.5	59.5		5.0	76.6	1.6	16.4	0.4	13,190				
Wyoming	Sheridan	Monarch	Monarch No. 45	SBB	1	23.2	31.2	41.7	3.9	6.5	55.1	1.1	32.9	0.5	9,540	63	32	4	1
					2		40.6	54.3	5.1	5.1	71.7	1.5	16.0	0.6	12,420				
					3		42.8	57.2		5.3	75.6	1.5	16.9	0.7	13,080				
North Dakota	Mercer	Ft. Union	Beulah[c]	Lig	1	34.9	26.0	31.1	8.0	6.6	41.8	0.6	42.0	1.0	7,000	64	20	14	2
					2		40.0	47.8	12.2	4.3	64.2	0.9	16.9	1.5	10,800				
					3		45.5	54.5		4.9	73.0	1.0	19.4	1.7	12,150				
North Dakota	Ward	Coteau	Velva[c]	Lig	1	39.5	26.0	30.0	4.5	7.1	40.3	0.6	47.1	0.4	6,820	61	27	9	3
					2		43.0	49.5	7.5	4.5	66.6	1.0	19.8	0.6	11,280				
					3		46.5	53.5		4.9	72.0	1.0	21.4	0.7	12,200				

[a] BHVA, Bituminous high-volatile A; BHVB, bituminous high-volatile B; BHVC, bituminous high-volatile C; SBB, subbituminous B; Lig, lignite.

[b] Condition 1, coal as received; condition 2, moisture-free; condition 3, moisture-and-ash-free.

[c] Name of town.

[d] This result was obtained on a column sample of the coal; the sample hydrogenated was washed and therefore much lower in fusain content.

yields given in Table 4. The average percentage gas and water yields for the high-volatile bituminous, subbituminous and lignitic groups were 20, 24, and 25; and 10, 13, and 15, respectively. The increase in yield of water with decrease in rank was not as large as would be expected if all the oxygen of the coal substance in excess of its hydrogen equivalent to form water had combined with hydrogen in the liquefaction process. It was shown in the research with the small autoclave that much of the oxygen of subbituminous coals and lignite is evolved as oxides of carbon, especially carbon dioxide. This conclusion is supported further by evidence from the continuous hydrogenation experiments as shown in the hydrogen consumption values of column 11 of Table 4. The lignites absorbed somewhat less hydrogen than the high-rank coals.

TABLE 4.—*Liquefaction Yields for Optimum Conditions of Hydrogenation*

Origin of Coal, Mine or Town and State	Percentage of Dry, Ash-free Coal					Oil Yield, Gal. per Ton Coal as Mined
	Hydrogen Absorbed	Oil Yield	Total Aromatic Neutral Oil Yield	Tar Acids Yield[a]	Tar Bases Yield	
1	11	12	13	14	15	16
Sayreton, Alabama	8	62	33	8	2.5	136
I.C.C. No. 21, West Virginia	11	69	35	5	2.0	156
Empire, Alabama	12	73	31	8	1.7	176
U. S. Experimental, Pennsylvania	9	72	41	11	2.8	169
Columbia, Utah	7	74	30	11	2.6	170
Orient No. 2, Illinois	8	71	37	15	1.7	155
Saxton No. 1, Indiana	9	70	32	12	2.3	141
Strain-Upper Diamond, Washington	8	70	28	15	3.7	155
Colstrip, Montana	8	63	26	14	2.1	108
Puritan, Colorado	10	63	28	13	2.8	115
Monarch No. 45, Wyoming	8	66	25	16	3.1	123
Beulah, N. Dakota	7	54	24	12	1.6	79
Velva, N. Dakota	7	55	22	15	1.6	79

[a] Distilling up to 330°C.

The lower rank coals, including lignite, subbituminous coal and high-volatile C bituminous coal, yielded from 12 to 16 per cent of tar acids on the basis of the dry, ash-free coal. The higher rank coals yielded from 5 to 15 per cent of tar acids. High-volatile B and C coals appear to give the largest yields of tar acids in pounds per ton of coal as mined or washed. The highest yields were 269 and 258 lb., respectively, from the Strain-Upper Diamond, Washington, and the Orient No. 2, Illinois, coals. The Wyoming subbituminous coal also gave the high yield of 219 lb. The best lignite yield was 168 lb. While these figures are subject to considerable variation with changes in operating conditions of the liquefaction process, they show that the products of coal hydrogenation constitute an important potential source of raw materials for plastics.

Column 13 of Table 4 shows that with a few exceptions the total aromatic neutral oil in percentage of dry, ash-free coal decreased as the rank of the coal decreased. These oils constitute about half of the total oil yield and they can serve as good stock for the preparation of high-octane aviation fuel.

Comparison of Results of Continuous Assay and Small Autoclave Tests, with Predictions from Chemical and Petrographic Analyses

Table 5 shows the relation between the dry, ash-free carbon content of the coals and the yields of oil from the continuous assay tests. It shows also the relation between the residue yields of the continuous assay and autoclave tests on the one hand and the predicted yields of residue on the other. These predictions are based on the assumption that all of the fusain and 40 per cent of the opaque attritus remains in the organic residue.

TABLE 5.—*Comparison of Results of Continuous Assay and Small Autoclave Tests with Predictions from Chemical and Petrographic Analysis*

Origin of Coal, Mine or Town and State	Rank of Coal[a]	Percentage of Dry, Ash-free Coal				
		Carbon Content	Yield of Assay Oil	Yield of Organic Residue		
				Predicted[b]	Continuous Assay Tests	Autoclave Tests
1	2	3	4	5	6	7
Sayreton, Alabama	BHVA	86.9	62	9.0	9.8	8.7
I.C.C. No. 21, West Virginia	BHVA	86.6	69	3.4	5.6	
Empire, Alabama	BHVA	84.4	73	7.8	6.2	
U.S. Experimental, Pennsylvania	BHVA	84.2	72	7.8	4.0	7.5
Columbia, Utah	BHVB	82.1	74	5.4	5.3	
Orient No. 5, Illinois	BHVB	80.6	71	5.4	4.9	6.7
Saxton, No. 1, Indiana	BHVC	80.9	70	6.8	6.0	
Strain-Upper Diamond, Washington	BHVC	78.6	70	0.4	1.1	3.7
Colstrip, Montana	SBB	76.7	63	8.2	6.7	7.4
Puritan, Colorado	SBB	76.6	63	8.0	5.6	8.3
Monarch No. 45, Wyoming	SBB	75.6	66	2.6	3.1	3.8
Beulah, North Dakota	Lig	73.0	54	7.6	9.7	5.0
Velva, North Dakota	Lig	72.0	55	6.6	4.7	4.8

[a] BHVA, bituminous high-volatile A; BHVB, bituminous high-volatile B; BHVC, bituminous high-volatile C; SBB, subbituminous B; Lig, lignite.

[b] Predicted by assuming that fusain and opaque attritus give 100-per cent and 40-per cent yields of organic residue on hydrogenation.

The lower oil yield of the Sayreton, Alabama, coal probably is due partly to its relatively high content of fusain and opaque attritus and to its high content of inorganic matter. On the other hand, the coal from Monarch, Wyo., gave a higher yield than its position in the carbon scale would indicate, and this difference may be attributed to its low content of insoluble petrographic constituents and its low ash. The oil yields of the other coals, with the exception of the resinous Utah coal, follow the order of the carbon content reasonably well considering the effect of operating variables in the hydrogenation process.

The organic residues from the continuous assay tests vary both above and below the predicted residues, the differences in most instances being between 2 above and 3 below the percentages of predicted residues. The average algebraic difference for nine comparisons was 1.5 per cent of the dry, ash-free coal. The residues from the continuous assay tend to be from zero to about 3 units below the results of the autoclave tests. It is evident that the degree of liquefaction of coal can be predicted almost as well from chemical analysis and petrographic examination as from the small autoclave tests. Either method may be used for the preliminary examination of coals with reference to determining their probable suitability for hydrogenation.

A very low percentage of fusain and opaque attritus when it accompanies a low percentage of ash (4 per cent or less) appears to favor complete and easy liquefaction. The high-volatile C bituminous coal from Washington and the Monarch mine subbituminous B coal from Wyoming were the most easily liquefied coals of any thus far tested in the continuous unit. The heavy-oil slurry had a low viscosity and was readily handled through the pumping system. It was possible to operate with relatively high rates of throughput. Coals containing high proportions of fusain, opaque attritus, and inorganic matter offer operating difficulties on account of high viscosity of the heavy-oil slurry and the formation of asphalt-like solids that are difficult to hydrogenate to liquids. Most of these difficulties can be overcome by suitable adjustment of operating conditions.

Characteristics of Oils Produced

The factors governing the characteristics of the oils produced in the liquid-phase hydrogenation of coals may be classified in two groups, the first being those related to the rank and type of coal and its chemical and petrographic composition, and the second being the operating variables such as the temperature, pressure, time of contact, catalyst and amount and temperature of refluxing of the overhead oil before it leaves the converter. A study of the results of the numerous different runs under various operating conditions for each of the 13 coals tested in this investigation showed that the operating variables, and especially the reflux temperature, had a much greater effect on the characteristics of the overhead oils than did the rank, type and composition of the coal. The only characteristic measurably influenced by the composition of the coal was the percentage of tar acids, which tended to increase as the rank decreased. The average percentages of tar acids by volume in the overhead oils were 15, 24, and 26, respectively, for high-volatile bituminous, subbituminous and lignitic coal. The percentages of olefins in the neutral oils showed a slight tendency to increase as the rank of coal decreased. However, these changes were much less in amount than was produced by changes in the reflux temperatures in the converter by different runs on the same coal.

In order to give a simple summary of the magnitude of the ranges of composition and characteristics of the overhead oils, the data from the various runs (excluding those made under unfavorable operating conditions) on each coal tested were averaged. The minimum, maximum and average of these averages are given in Tables 6 and 7. The distillation data in Table 6 show that an average of 27 per cent of the overhead oil distills in the gasoline range, up to 392°F. (200°C.), 50 per cent between 392° and 572°F. (200° and 300°C.), and 23 per cent above 572°F. (300°C.). The specific gravities of the overhead oil and of the various fractions are higher than the corresponding petroleum products, owing

TABLE 6.—*Specific Gravity and Distillation Analysis of Overhead[a] Oils from the Liquid-phase Hydrogenation of Coals*

Distillation Range		Percentage by Volume of Overhead Oil			Sepcific Gravity at 60°F. (15.6°C.)		
Deg. F.	Deg. C.	Minimum	Maximum	Average	Minimum	Maximum	Average
1	2	3	4	5	6	7	8
68–392	20–200	20	31	27	0.81	0.88	0.83
392–518	200–270	27	42	34	0.94	0.97	0.95
518–572	270–300	13	18	16	0.96	0.99	0.98
572–626	300–330	9	17	14	0.98	1.01	0.99
Over 626	Over 330	4	13	9			
Total overhead oil.....					0.91	0.96	0.93

[a] Overhead oil is that which is obtained by condensation of the oil vapors carried out of the top of the converter by the hydrogen gas stream.

to the presence of large proportions of aromatic hydrocarbons, tar acids and tar bases. From one third to two thirds of the tar acids is phenol, cresols and xylenols. A distillation analysis of the tar acids from the hydrogenation of coal from the U. S. Experimental Mine indicated the following composition, in percentage by volume of total tar acids: phenol, 8; cresols, 27; xylenols, 30; and higher tar acids, 35. These are important raw materials for the manufacture of plastics.

TABLE 7.—*Composition of Overhead Oils*[a] *from the Hydrogenation of Coals*

Percentage by Volume of Overhead Oil	Minimum	Maximum	Average
Tar bases	2	5	3
Tar acids	8	29	20
Neutral oils	68	89	77
			100
Specific gravity of neutral oils at 60°F. (15.6°C.)	0.88	0.96	0.91
Percentage by Volume of Neutral Oils			
Olefins	6	13	10
Aromatics	46	73	54
Saturates	20	44	36
			100

[a] Overhead oil is that which is obtained by condensation of the oil vapors carried out of the top of the converter by the hydrogen gas stream.

The principal constituent of all the fractions of overhead oil was neutral oil. While the analytical determination of the olefins, aromatics and saturates is not as accurate as could be desired, the composition given in Table 7 gives a good approximation of the chemical nature of the neutral oils. The high average of aromatics (minus 54 per cent) indicated a reasonably good octane value for gasoline made from these oils. The percentages of olefins and saturates averaged 10 and 36, respectively. The saturates consist of approximately equal parts of naphthenes and paraffins. The average approximate amounts of some of the hydrocarbons found in the neutral oils boiling below 392°F. (200°C.) are given in Table 8.

The average fraction of neutral oil boiling below 392°F. (200°C.) is about 27 per cent of the total oil yield and the neutral oil averages 77 per cent of the total overhead oils or oil yield from the coal. On the basis of these values as given in the last column of Table 8, 0.5 gal. of benzene, 0.6 gal. of toluene, and 2.1 gal. of xylene could be obtained from a ton of bituminous coal as mined, yielding 160 gal. of oil.

TABLE 8.—*Average Approximate Amounts of Some Hydrocarbons Found in the Neutral Oils Boiling below 392°F. (200°C.) from Liquid-phase Hydrogenation of Coals*

Hydrocarbons	Percentage by Volume[a]	
	Of Neutral Oils	Of Overhead Oils
Benzene	1.5	0.3
Toluene	2.1	0.4
Xylene	6.3	1.3
Cyclohexane plus methylcyclohexane	6.1	1.2
Dimethylcyclohexane	4.5	0.9
Tetrahydronaphthalene	13.9	2.8
High-flash naphtha or polymethyl benzenes	65.6	13.1

[a] Assuming that the neutral oil averages 77 per cent of the overhead oil or oil yield and that the fraction boiling up to 392°F. (200°C.) averages 27 per cent of the overhead oil, the neutral oil boiling up to 392°F. is 0.77 × 27 = 20.9, or 20 per cent, in round numbers.

From these data, it is evident that commercial quantities of chemical products of higher value than liquid fuel can be obtained in addition to the fuel. This conclusion applies in particular to the tar acids, phenol, cresols, and xylenols.

SUMMARY AND CONCLUSIONS

In concluding this brief review of the Bureau of Mines research on the direct hydrogenation and liquefaction of American coals, it must be emphasized that no pilot-plant investigations involving the use of commercial-scale units, or even approaching commercial-scale units, have yet been made. The next step the Bureau of Mines desires to undertake is the construction and operation of a pilot plant on a considerably larger scale, in which the size of the converter unit is near enough to that of a commercial plant so that the results can be used for the design of commercial plants. The present converter is only 3 in. inside diameter, whereas the converters at Billingham, England, are approximately 5 ft. inside diameter. These differences are too great for direct interpretation of results from one to the other.

The investigation thus far made by the Bureau has demonstrated that the major part of the American reserves of coal are suitable for liquefaction. This part includes the common bright coals of high-volatile bituminous, subbituminous, and lignitic rank. Coals low in ash and nearly free from fusain and opaque attritus were almost completely liquefied. The most favorable operating conditions for the maximum yield of oil vary with different coals; however, the physical characteristics and chemical composition of the resulting oil are influenced principally by the operating conditions. The highest yields of phenols and tar acids on the basis of coal as mined are obtained from high-volatile B and C bituminous coals, and possibly from subbituminous A coals also.

These tar acids occur in sufficiently large amounts to be commercially important as sources of raw materials for the growing plastics industry. The high content of aromatic compounds in the neutral oil makes it a desirable blending base for high-octane aviation gasoline. It is possible also to modify operating conditions so as to produce principally fuel oil, or fuel oil, diesel oil, and gasoline.

Further work in coal hydrogenation should include both laboratory and pilot-plant research. No one can predict the exact time when these processes can operate in this country on a commercial basis without subsidy, but in the meantime it does appear desirable in the national interest to be fully informed on the possibilities of supplementary sources of liquid fuels, so as to be prepared to utilize these sources when the need occurs.

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DISCUSSION

(H. H. Lowry presiding)

A. R. POWELL,* Pittsburgh, Pa.—The volume of high-quality research work on coal hydrogenation carried on by the Bureau of Mines, particularly since 1936, can be realized only after one has read the comprehensive publications and has listened to some of the hearings before the Senate subcommittee considering synthetic liquid fuels, and also has visited the laboratories and small-scale plants at the Pittsburgh station of the Bureau.

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One phase of this coal-hydrogenation work intrigues me very much. This is the long-shot possibility of hydrogenating and liquefying coal or lignite without using any outside hydrogen. The authors have given some hints that there are possibilities of at least partly reaching such an objective.

As pointed out in the paper, coal has a ratio of hydrogen atoms to carbon atoms of about 0.83, whereas petroleum has double the relative amount of hydrogen, or a ratio of 1.66. The approach to such an increased ratio of hydrogen to carbon may be obtained not only by addition of hydrogen but also by subtraction of carbon. The following three possibilities are suggested by the authors' paper:

1. Since oxygen must be eliminated from the coal during liquefaction, it is advantageous to have it eliminated as carbon dioxide, since this reduces the amount of carbon. The authors have pointed out that this occurs to the maximum extent in lignite and low-rank coals, and, since the oxygen content of these coals is high, some interesting possibilities are presented. Unfortunately, even complete removal of the oxygen from the moisture-free coal in the form of carbon dioxide would not secure the desired hydrogen-carbon ratio, but at least it helps. The authors indicate that 80 per cent of the oxygen in lignite is of the so-called "active" form, which should be favorable provided the correct conditions were found for optimum carbon dioxide formation.

2. Water gas formation. In *Technical Paper* 654, page 23, Dr. Storch et al. mention that at 430°C. the water-gas reaction is superimposed upon the thermal decomposition of the coal substance. This is interesting because the water-gas reaction not only subtracts carbon by forming carbon oxides but also adds to the hydrogen available, so that approach to the desired hydrogen-carbon ratio is approached from two directions. Do the authors consider this possible autogenous liquefaction of coal without the use of outside hydrogen a promising field of research?

3. Partial liquefaction of coal. The question naturally arises as to the desirability of *completely* liquefying coal. The authors state that almost complete liquefaction and extensive oxygen removal can be obtained at comparatively low temperatures and with little utilization of hydrogen. The authors then point out that this is largely due to the reservoir of hydrogen present in the hydroaromatic liquid serving as a vehicle, and this hydrogen must, of course, be replaced in the vehicle later. However, is it not possible to secure a limited quota of liquefaction without supplying hydrogen? Would it not be more economical to accept whatever liquefaction may be obtained easily and then run, leaving the solid residue for those fuel uses for which solid fuels are best adapted?

The Bureau of Mines, and particularly the authors of this paper, are to be highly commended for their foresight in initiating this program of research many years before public opinion in general believed that our country should provide for future dwindling production of petroleum. The wisdom of this course is widely recognized today. We trust that this program will be expanded in the near future to the construction and operation of demonstration plants, so that we in America will have the "know how" of engineering design and economics for our own future use in our country.

H. H. Storch (author's reply).—In obtaining a heavy fuel oil of bunker C type by the Bergius-I.G. process, our tests in the experimental plant show that about 5 per cent by weight of the coal (ash- and-moisture-free) of hydrogen is consumed. To obtain gasoline, about 11 per cent of hydrogen is necessary. In our experimental plant, as in the Bergius plants in Germany and Great Britain, the hydrogen is supplied as gas at 200 to 300 atmospheres pressure.

The water-gas reaction is not fast enough at 450°C. under the conditions existing in coal hydrogenation to supply the necessary hydrogen, by reaction of steam with the coal. For this reason, very considerable amounts of coke are produced when only water and coal-oil paste, but no hydrogen, are used. It is conceivable, of course, that catalysts may be discovered that will accelerate the reaction between water and coal at 450°C. to such a degree that the hydrogen will be produced at least as rapidly as the decomposition products of the coal are formed. At the present time this possibility appears quite remote.

A. C. **Fieldner** (author's reply).—It is possible to secure a part of the hydrogen necessary for liquefaction from the gases produced. The hydrocarbons such as methane, ethane, and propane can be reformed with steam to yield hydrogen for use in the process. However, this step is subsequent to hydrogenation.

It has been suggested to us that the liquefaction of coal might be conducted in conjunction with the manufacture of fuel gas. For example, in western Pennsylvania and southeastern Ohio the natural gas supply is decreasing and there will be a need of manufacturing fuel gas from coal and introducing it into the pipe lines traversing this region to satisfy the requirements of the industries and cities served by these gas-transmission lines.

F. H. **Reed**,* Urbana, Ill.—There is a tremendous amount of work reported in this paper—twenty years research at least by the Bureau of Mines. The thing that has appealed most to some of us that have gone over it is the exceedingly fine way in which the research has been carried on; that is, a small scale, a second scale, a little larger, and now a contemplated experimental scale, which is still larger.

There is no question but that this research costs a tremendous amount of money, but there is also no question but that this is the cheapest way to go at it.

One question I would like to ask in regard to the by-products that are formed from this, particularly those that may be used in the chemical industry. It is pointed out in the paper that the high oxygen compounds naturally give more of the tar acids, phenol, cresol, xylenol, etc. There are also the gas by-products.

In reading this paper, also in reviewing the hearings that have been held in regard to the advisability of larger scale experimentation, and in reading other reviews, we find estimates of cost of gasoline and fuels by hydrogenation of coals from other sources, particularly our shales. In estimating those costs the question arises: How much consideration has been given and how much can we give to the evaluation of these by-products. It would seem to me that, as in the coking of coal, what we can get for the by-products may determine the cost of production of the liquid fuel.

A. C. **Fieldner**.—We have no figures on probable returns from the by-products, and there are no published figures available on foreign work. The principal by-products are the phenols, cresols, xylenols and higher tar acids. They comprise about 15 per cent of the crude liquid product. These can be separated and sold as chemical products.

Dr. Storch is enthusiastic over the possibilities of utilizing them in the plastics and chemical industries. Benzol and toluol also can be separated or they may be left in the motor fuel fraction to improve its octane value and utilization in aviation fuels. Such aviation fuels, of course, sell for a higher price than motor gasoline.

Perhaps I should have given greater emphasis to the fact that the work reported in this paper relates only to the Bergius-I.G. hydrogenation process. There is another process for making synthetic gasoline known as the Fischer-Tropsch process, in which coal or coke is converted to hydrogen and carbon monoxide by the water-gas process. These gases are then converted catalytically to liquid hydrocarbons. The Bureau is working on this process also and will report on it at a later date.

The Fischer-Tropsch process can be applied to any raw material from which water gas can be made. This includes coal, anthracite coke, lignite, and methane or natural gas. Natural gas at three to five cents per thousand cubic feet probably is the cheapest source of synthetic gasoline. The nature of the liquid products from the Fischer-Tropsch process is different from that of the products from the Bergius process. They are aliphatic compounds and must be reformed to raise their octane value.

I believe that methods will be worked out for making good aviation gasoline by gas-synthesis methods.

There is some difference in the gasolines obtained from different coals by the Bergius hydrogenation process, but that difference is much less than the differences due to variation in operating conditions. We can vary the operating conditions and get quite different characteristics in the liquid fuels produced. It is not at all certain that our low-rank subbituminous and lignitic coals will prove to be the cheapest sources for gasoline. It may develop that we can make synthetic gasoline more economically from bituminous coals.

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* Illinois State Geological Survey.

First, the yield from bituminous coals per ton as mined is twice as large as that from lignite; second, they are nearer the markets. On the other hand, from a long-time point of view, we should conserve our best coking coals, of which our reserves are limited, and not use them for making liquid fuel. Instead, we should liquefy low-rank coals not suitable for making coke.

H. I. SMITH,* Washington, D. C.—What part of a ton of coal would it take for power and other purposes to liquefy a ton of coal?

A. C. FIELDNER.—The British experience at Billingham indicates that it takes about as much coal for the production of additional hydrogen, steam and power as the amount of coal liquefied. The gross liquefaction yields should be divided by two to obtain the net yield per ton of total coal used. In other words, one ton of coal is required for power and other purposes to liquefy a ton of coal.

H. I. SMITH.—These figures, then, were on the total coal consumption?

A. C. FIELDNER.—No, they represent the yields per ton of coal passed into the liquefaction converter and are approximately twice the yields per ton based on the total coal used for all purposes in the liquefaction plant.

R. SHERMAN,* Columbus, Ohio.—I have one question, but Dr. Fieldner has at least partly answered it in his presentation. This is in regard to the hydrocarbon gases formed in hydrogenation. The volume of these gases produced is usually given in the discussion of the results of laboratory-scale hydrogenation experiments, but I have seen only one mention of economic uses of the gases.

One exception is in the 1935 description of the Billingham plant. At that time, it was stated that the gases were being burned under the boilers. This seems to be a wasteful use of these gases of high calorific value. T. H. Kerr, fuel consultant, and Dean MacQuigg, of the Ohio State Engineering Experiment Station, have suggested the possibility of the use of the gases to supplement natural gas supplies. Rough calculations indicate that the amount of gas produced is enough to constitute a very important credit to the process and perhaps make the hydrogenation of coal approach economic feasibility somewhat closer than it appears at present.

Dr. Powell has suggested self-hydrogenation of coal, with the use of the residual coal as fuel in conventional equipment. One fault with this would appear to be that the residual coal would have a high ash content. If the coal is washed to 2 per cent ash before hydrogenation and 50 per cent is recovered, the resultant 4 per cent ash coal would be fine, but if we are to start with normal coals of 5 per cent ash or higher, the resultant fuel would not be very acceptable. Equipment manufacturers are today asking the coal industry to reduce the ash content.

A. C. FIELDNER.—It is possible to partly hydrogenate coal and make a coking coal from a noncoking coal. This has been done by the Fuel Research Station in Great Britain.

As Mr. Sherman says, there are some possibilities of integrating liquefaction with gasification of coal. These are difficult to evaluate at the present time because we have not enough facts. The practice at Billingham is governed to some extent by special conditions. It is really a demonstration or commercial-scale pilot plant that was erected in connection with a synthetic ammonia and chemical plant.

M. B. COOKE,* New York, N. Y.—I have been doing some research work on the production of solid smokeless fuels. The indications are that the Germans are using the Lurgi process. They make the low-temperature tar in the Lurgi process, and hydrogenate the tar rather than coal because of the difficulties they have encountered with coal; the same that have been found in England. It is hard to get solids through pipes and valves under 8000 lb. pressure, without eroding and corroding them out. Also, it is possible to get about three times the yield of gasoline per day from the same unit. Dr. Fieldner brought out something like that.

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* U. S. Geological Survey.
* Battelle Memorial Institute.
* Consultant, American Lurgi Corporation.

(*Discussion continued on page 375*)

Research for the Coal Industry

By C. E. Lesher,* Member A.I.M.E.

(New York Meeting, February 1944)

Coal has been fighting a rear-guard action since the last World War. The battle against competitive fuels has been largely guerilla warfare with more sniping within the ranks than of organized opposition. Oil and natural gas are the fuels that over these twenty odd years have driven coal backward and out of many profitable markets.

Now, somewhat timidly, the coal industry is trying a new weapon; namely, Research, or let us more properly say, Organized Research. Engineers have been designing this weapon for many years, supported by those interested in coal, including some coal producers. A part of this pioneer work has been Fundamental Research, the Ph.D. kind that is over the heads of most of us. Another part has been aimed at immediate objectives and realizable profits, the practical, man-in-the-street sort. That is Industrial Research. Enough has been accomplished to make a fair exhibit, but whether this new weapon will have sufficient fire power to rout the opposition, whether the industry will really go into production and use it, the next few years will tell.

Need for Coal Research

To utter "Research" is to use the modern magic word; to conjure up the miraculous—rubber from weeds; radar; magnesium from sea water; yes, even gasoline from coal. Research is a very general term. It means something different to every man. What is the coal industry seeking with its research; or what should it seek? And, specifically, how is it going about the job; and is it taking the best way?

There are many who say that Coal has been asleep at the switch, while Oil and Gas have been taking the energy business. In an organized way this is true. Large-scale, privately financed research is just started in the Pennsylvania anthracite industry, and it is still a dream for bituminous coal. What has been done so far collectively is purely defensive; as, for instance, to improve mechanical devices for burning coal so as to be able to recover markets for house-heating fuel long since abandoned to oil and gas.

To appreciate just how seriously bituminous coal has slipped back in the national economy in the past 25 years, one has only to recall that, in the last war the Navy required for its operations all the high-grade, low-volatile coal that could be produced, and that anthracite was withdrawn from the middle and northwest markets to keep the eastern seaboard warm. All coal was "rationed" by zones, if not to individuals. Now the Navy, the Armed forces and the Merchant Marine are propelled by oil and gasoline. Gasoline and heating oil are rationed to individuals. Production of all coal in this country in 1918 reached the historic figure of 678,200,000 tons, and in 1943, after 25 years, it is expected to reach a special wartime total of 660,000,000 tons. Meanwhile, production of crude oil this year may exceed 1,500,000,000 bbl., and like that of natural gas, reach four times the amount produced in 1918.

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* President, Pittsburgh Coal Carbonization Co., Pittsburgh, Pa.

Coal wants to regain these lost markets—millions of tons that have been replaced by liquid and gaseous fuels in the past 20 years. In the early 20's the coal producers organized to fight foreign oil with import duties. Pipe lines for oil and for gas, designed to put these fuels into new markets, were and still are opposed with vigor. But the coal industry, torn by labor strife and internal friction, with no defense against the overwhelming tide of oil and natural gas, has throughout these years thrown its precious reserves of high-grade coal into the battle, at ruinous prices, and emerged an acknowledged pauper industry.

Battered and beaten in the competitive struggle with oil and gas, the coal industry is instinctively putting its research money on the biggest stake of all—tonnage. That is as it should be. Coal is energy—heat and power. The best markets are the heat and power consumers, who in large numbers ceased to use coal because they got more satisfaction for their fuel dollars with oil or gas.

Tonnage is all important to coal. With a production of six hundred million tons a year, coal is the largest material-handling job in the country. Take, for instance, the year 1940, when all coal produced was 505,000,000 tons. All the ores of iron, gold, silver, lead, zinc and copper mined in that year amounted to less than 186,000,000 tons. Or take, if you will, the nonmetallic mining and quarrying industries. Include sand and gravel with 238,000,000 tons, stone with 153,000,000 tons, add in cement, slate, salt, lime, clay, gypsum and a dozen others, and you have a total of less than 450,000,000 tons mined or quarried. Coal, most of it, is undercut, blasted, and loaded in small lots from thousands of "working places" far underground. Of the 746,000 men reported in 1940 as engaged in mining and quarrying in the United States, 532,000 were mining coal.

Hence, the industry that mines coal by the acre and ships it in railroad trains is tonnage minded. After petroleum has taken from coal more than a hundred million annual tons of this heating business, coal diggers may be expected to concentrate on ways and means of again filling those coal bins not already painted and furnished for living quarters and "game" rooms.

Present Cost of Coal Research

A conservative estimate of expenditures for bituminous-coal research in 1943 adds up to nearly seven hundred thousand dollars. The Bureau of Mines leads off with $400,000, which is exclusive of appropriations for such as mine safety and explosives. Most people do not realize that this Bureau has been spending a quarter of a million dollars annually on coal research for the past 20 years.

The work at Battelle Memorial Institute, financed by the coal operators and their associated sponsors, the railroads, has cost around $80,000 a year. The Bituminous Coal Research Laboratory at Carnegie Institute of Technology spends $75,000 a year, less than half of which comes from coal producers. The industry contributes to the fund of $30,000 for coal research at Pennsylvania State College. Illinois spends about $100,000 a year at Urbana, largely on studies of Illinois coal. Of the total, however, not over 20 per cent comes directly from the producers of bituminous coal. In past years the bituminous-coal industry assumed an even smaller share in the cost of research devoted to its interest, but it now promises soon to exceed all records. Since the last war, anthracite has suffered greater losses of tonnage proportionately than has bituminous coal, but it has done more about it.

It is true that in the past decade coal research has been more fundamental and less industrial than many would have wished, but that has not been the fault of the Federal bureaus or the universities where the work has been done with public funds. It is unfair to belittle the progress that has been made or to say that this country by comparison with foreign countries has been backward, or niggardly, in its approach to the problems of coal research.

RESEARCH OBJECTIVES

Let us define the broad objectives of coal research as the increased and more efficient utilization of solid fuels, and let us recognize that the effort will be profitable to the coal producer only if that greater utilization is in the public interest. The individual producer is the one who mines whatever coal is in his acreage. His first concern is cost on the cars; his problem: mining, safety, and preparation. The individual who stumbles in this mine-cost and quality race has always been trampled underfoot by his coal-producing colleagues. The industry loses to the competition of other fuels. Everything in engineering or management that contributes to safety, lower costs and better quality of product is a subject for research whereby to increase the market for coal on an industry-wide basis.

Such problems are not for research with the laboratory test tube, however. They are mining—cutting, loading, hauling, cleaning—all problems of efficient utilization of men, machines, power and explosives. These are the basic, primary problems of this industry, calling for continuing research and development without which there can be no greater tonnage. It is not proposed here to detail the marvelous results, particularly of the past decade, that have flowed, in spots only it is true, from research in mining and preparing coal. The mining industry has borne the cost of this research and development—for the most part indirectly, of course—in the price it pays for improved cutting and loading machines, drills, motors, track and trolley, explosives, jigs, screens and a thousand other items. The research of equipment manufactures, coupled with the willingness of mine managers and engineers to try new things, has made possible the lower cost and better quality of coal without which much of the industry today would indeed be in desperate condition. No discussion of research in coal can fairly overlook or fail to recognize, or refrain from acclaiming, the progress that has been made in mining and in preparing coal for market.

Progress in utilization of coal in the past decade certainly calls for no apology. The figures have become almost commonplace. In 25 years a decrease of 34 per cent in coal required by the railroads for hauling 1000 gross ton-miles of freight; a decrease of more than 50 per cent in the coal required to produce a kilowatt of electrical energy by central power stations; 20 per cent less coal to make a ton of pig iron. These are cheering figures, for which the combined research and development by manufacturers of equipment, power-plant engineers, coke-oven and blast-furnace operators—the consumers—are mainly responsible.

The sum total result of the efforts of combustion and carbonization technicians and engineers to put the proper coal to each individual use, to see that it is better, cleaner coal, to make each B.t.u. do full duty, and to conserve the by-products, is that coal is now used more efficiently than ever before. The coal industry has not lost tonnage because it has been party to increasing combustion efficiency; on the contrary, it has held tonnage it most certainly would otherwise have lost to its competitors.

HISTORY OF COAL RESEARCH

Coal research in this country has an honorable past. Forty years ago the United States Geological Survey opened its Coal Testing Plant at the St. Louis World Exposition of 1904. Those who pioneered this work on coal combustion and boiler efficiency set a high-water mark for coal research. Not the least of its accomplishments was that from that effort was born the Bureau of Mines in 1910.

There was a worldwide revival of interest in coal research immediately after the first World War, and the recrudescence of 1943 is not dissociated from the war effort and present ability of the coal industry to finance on its part, an enlarged program.

Fundamental research was given its greatest impetus in this country, and high-lighted throughout the world by the historic International Bituminous Coal Conferences, organized by the late Dr. Thomas S. Baker, then President of Carnegie Institute of Technology. The first of these, in 1926, was a forum for the international celebrities of coal chemistry and research. Succeeding meetings in 1928 and 1931 greatly enriched the literature and promoted the exchange of ideas on this important subject. It is interesting to look back in the record, to reconsider the objectives of coal research as set forth then and more recently, and then to pause for a short look ahead into the future of power and of the chemical age, with perhaps some speculation as to the part that coal may play, both as the primary source of energy as well as a raw material for the chemical industry.

At the Coal Conference in 1926, John Hays Hammond and F. G. Tryon[1] paid tribute to fundamental research, cautioned against overemphasis on the practical approach before the fundamentals were established, and urged that economic problems be not overlooked in any research program for the coal industry. Economic problems of this industry were then, and still are, overpowering factors in its prosperity and progress, but the advice was not followed, for the industry has provided no organized economic study or research.

Dr. A. C. Fieldner,[2] at the conference in 1926, presented a concrete program for coal research, based on what had already been inaugurated for petroleum. Of 30 projects on which work was started by the American Petroleum Institute, covering geology, physics and chemistry of petroleum, Dr. Fieldner found that 22, involving fundamental research, had direct and significant counterparts for coal. A year later, Dr. Fieldner[3] analyzed the results of a worldwide inquiry into coal research conducted by the National Coal Association, and suggested as subjects for investigation:

- Constitution of Coal and Coke
- Mining, Safety and Health
- Preparation and Storage
- Combustion
- Processing
- Industrial Analysis and Testing

These titles are still good.

On July 1, 1930, the Coal Research Laboratory of the Carnegie Institute of Technology was organized under the directorship of Dr. H. H. Lowry, sponsored by an impressive array of corporations, including sixteen coal producers, five railroads, six of the largest industrials, the Buhl Foundation, Carnegie Institute of Technology and the National Coal Association. One individual name ends the published list of financial sponsors, Howard N. Eavenson. This laboratory, financed on a broad base, had fundamental coal research for its purpose. Its work was barely started when the third, and last, of Dr. Baker's International Bituminous Coal Conferences was held at Pittsburgh in 1931. At the Second Coal Conference, in 1928, Mr. Eavenson[4] forecast the basis for the organization and support of this laboratory, and it is due to his unselfish and untiring effort, as well as that of Dr. Baker, until his death in 1939, that the project has continued to receive adequate financial support.

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[1] References are at the end of the paper.

During the past 10 years the subject of coal research has certainly had its fair turn at coal conferences and technical society meetings wherever coal engineers and the scientifically minded have assembled. Stephen Burke[5] aptly divided the field when he differentiated "Research on Coal" from "Research for the Coal Industry." The "development of a product or process using coal as a raw material," he stated, "is research on coal," whereas to "widen the coal market until it more nearly coincides with the entire energy market, is research for the coal industry."

Dr. Lowry[6] has pointed out that the national waste involved in smoke, a nuisance associated in the public mind with bituminous coal, is the first point in a research program. He advised that coal either should be burned smokelessly by means of proper mechanical equipment or converted to smokeless fuel, which, since 1.4 tons of coal is required to make a ton of coke, would help tonnage. Processing, he warned, will be economically possible when the maximum return is obtained for the by-products. The principal use of coal, Dr. Lowry notes, is for combustion in the raw state, which represents 85 per cent of the normal market. Carbonization accounts for the remaining 15 per cent. Hydrogenation eventually may be a third use. In pointing out, in 1936,[7] the importance of development of small stokers as the only chance to regain markets lost to oil and gas, he defined the course that has been followed by the industrial research group.

The great advantage of coal, according to E. G. Bailey,[8] lies in its relatively lower cost per B.t.u. He finds that the characteristics of coal that make it less desirable than oil or gas are its "nonfluidity, size, igniteability (or lack of it), agglutinating property, volatile matter and ash content," and he suggests that to make coal as equally convenient as gas or oil may be too costly. "Why not improve cleanliness and convenience in the use of coal without materially increasing the cost of preparing it or the cost of burning it?" Mr. Bailey has repeatedly urged that the coal companies should carry on coal research in connection with small units where the cost of equipment and the amount of coal required for development and test purposes is not large.

The advice of Mr. Bailey[9] to the coal operators at Cleveland in October 1943, on the occasion of launching the drive for "A New Five-Year Research Program,"[10] is to concentrate on equipment for house heating and installations using up to 5000 lb. of coal an hour, and to stay in the cheering section where large power plants, pulverized-coal burning, new locomotives, processing and chemicals from coal are concerned. He is willing to leave hydrogenation and coal-burning turbines to the future.

ORGANIZATION OF THE INDUSTRY FOR RESEARCH

It is inherent in the structure of the bituminous-coal industry that large-scale research or development projects can be financed best on a cooperative basis. A research project is an expedition into the unknown, from which no one can guarantee a safe and profitable return. Research takes time—years of costly, time-consuming effort, no matter in whose hands it be entrusted. This industry has too few, if indeed it has any, units of sufficient size and power to embark on such large-scale programs as commonly are recognized as essential and worth while.

Cooperative industrial research has not been conducted on a large scale in this country, however. It was tried and abandoned a quarter century ago by the petroleum industry. Today petroleum research leads the field, and it is by each company with its own staff and laboratories. These are large corporations, very large as compared with the largest of the coal companies. The steel industry is now deep in research, having been almost as laggard as coal, and each steel producer is doing his own delving into alloys and eutectic mixtures.

The chemical industry does research in its own back yard on a stupendous scale, as does the American Telephone and Telegraph Co. on communications, the Radio Corporation on electronics, the Koppers Company on coal carbonization and processing. The list is long, and there are a few but not too important exceptions. On the other hand, industry often bands together in trade associations and institutes where economic and political factors are sufficiently compelling.

Both anthracite and bituminous-coal industries are experimenting with cooperative organizations engaged in technical research. Support by the industry for the Coal Research Laboratory at Carnegie Institute of Technology has been limited to a few strong companies, but even there it has required genuine leadership to maintain and strengthen it over its 13 years of increasing usefulness. Dr. Lowry set out to vivisect the coal molecule, and I suspect his sponsors have given him a free hand. Fundamental research is not presumed to produce marketable machines. It is the raw material, the basic data on which development research is based. Because of this, collective support of such laboratories is feasible, in industry as in government.

The story is different, however, with respect to collective efforts searching for, let us say, new machines designed to recapture lost markets for coal, or seeking ways and means of dressing up this fuel for acceptance in the same category of ease of handling and of cleanliness as its persistent competitors. The story is different because one not trained in physical chemistry must needs take the work of the fundamental researcher on faith, but the coal producer who also supports industrial research expects results that he may understand and that will benefit and profit, if not his individual enterprise, then most certainly his industry. The position is clear, however. In this industry, the only practical solution is collective research effort, with all its problems and difficulties. Its supporters must also take a lot on faith. If you buy shares of General Motors Corporation, you are content in the knowledge that the organization is backed by research, and, what is most important, by a competent sales organization to put over the things developed in the laboratory.

It is here that our coal research program is to have its real test. Bituminous Coal Research, Incorporated,[10] has developed "principles for smokeless heating stoves and ranges of new design," and the "correct principles for design of over-fire air jets." Where is the catalyst that will engineer, market, merchandise and make profitable these principles?

After one laboratory has established basic principles, and another has found a way to use these concepts, there still remains the final laboratory test of manufacture, selling and public acceptance. That is why no one must expect too much too soon from organized bituminous-coal research. Mr. Eavenson,[11] President of Bituminous Coal Research, Inc., notes that several projects have reached a stage where research work has ended and commercial development should begin; but, he says, "This is not the function of Bituminous Coal Research, and the industry should provide some organization to push the development of their projects."

Bituminous Coal Research, Inc., proposes to multiply its budget by 5 and to spend $2,500,000 in 5 years, which is really a modest program for an industry as large as this, but a program that will tax the energy of all who are to expend the money. A program[12] that includes investigations in 12 principal divisions of coal technology, and lists more than 90 specific projects that should be undertaken, will surely ring the bell on a few good ones. One should not be critical because the advance literature on the two and a half million dollar program is sparkling with "possibilities" for coal in new uses, for without imagination there can be no research, and a researcher who does not secretly believe that he will next day, maybe, accomplish a miracle, is not worth his salt as a researcher.

WHAT RESEARCH PROMISES

Charles F. Kettering,[13] who directs research for General Motors, has said:

> We must treat research as an insurance policy. Whatever we pay for research we must think of on the actuarial basis. We can consider a large number of projects and arrive at an average cost. When we take out a life insurance policy, nobody tries to figure out when we are going to cross the mortality line. Some cross below; some above. Likewise, in the case of research work; for some who run off the track and fail to accomplish anything new, we can expect a few to meet with success.

Business men and some professors, according to L. V. Redman,[14] "appear to have the impression that knowledge gained in research, if of potential value in industry, is ordinarily capable of immediate practical application." Research proper, he says, is relatively inexpensive, but the industrial exploitation of research, or development, may be a costly procedure. Redman has projected the course of a research project and finds that after three years of uncertainty and doubt, a project may look as though it would pay, and if all goes well, the enterprise should after five years be on a satisfactory paying basis. He notes that "The duration of this period of suspense, the total cost of development, and the chance of ultimate success will depend probably more on the thoroughness of the initial research and development than on present resourcefulness."

A great many coal producers were mildly excited by Secretary of the Interior Ickes,[15] who is also Petroleum Administrator and Solid Fuels Coordinator for War, when, in the spring of 1943, he said:

> A new industry based on the use of coal looms on the not too distant horizon. Outstanding among drastic changes the present world conflict is bringing about in our way of living is the use of coal on a scale and for purposes never before seriously considered, even in its heyday as the nation's overwhelming preponderant source of fuel and energy. The use of coal for new purposes and in ever greater quantities should vary our economic pattern. It scarcely can fail to increase the size and importance of many industries while creating new and powerful ones.

This startling prediction, it developed later, was a curtain raiser for a drive to get a considerable appropriation from Congress for building a coal-hydrogenation plant. Whether we should anticipate an oil famine in 15 years, as Secretary Ickes fears, and whether the Government should now plunge into coal hydrogenation to replace gasoline for our automobiles, are matters outside the scope of this paper. Behind the Secretary's bogey man there is a fundamental truth—that coal is a raw material for the chemical industry.

Already there are more chemicals, drugs, dye stuffs, plastics, explosives, and disinfectants made from what are commonly called "coal tar products" than are made from petroleum. Furthermore, the quantities are greater. But neither need be true for long, because research in petroleum laboratories and developments in oil cracking and refining are proceeding at a furious pace. High-powered chemistry is supplying the war effort with toluene, aviation gasoline and synthetic rubber from petroleum and will most certainly follow with benzol and acetylene as it is doing with cresylic acid.

Coal has been the source of the chemicals from which the greater part of plastics—the phenol formaldehyde resins—have been made. Petroleum and natural gas are challenging coal in this expanding field. R. L. Wakeman and B. H. Weil,[16] in a paper before the American Chemical Society in October 1942, summarize the place of coal as a source material for the plastics industry in these words:

> Coal is thus an important source material for the plastics industry. Plastics made from coal intermediates have shown each year a healthy rate of increased production. It seems proper to envision much greater expansion after the war. But the coal industry must take cognizance here, as in other chemical industries which use its products, of the recent but already important factor of competition with petroleum and natural gas hydrocarbon derivatives. . . . The coal industry's domination of the phenolic resin field may also soon be challenged not only by phenol from petroleum benzene, but likewise by increasing quantities of petroleum cresylic acids. . . . The coal industry need not necessarily suffer from this competition. . . . The prospects for a tremendous increase in the volume of plastics production will render the future for coal derivatives extremely bright if every effort is made to maintain research, reduce basic costs of manufacture, increase yields, and produce a varied and flexible number of intermediates.

Neither these words nor those of Secretary Ickes were intended for coal producers. Obviously they are addressed to the chemical industry. The coal industry does not make phenolic resins, acetylene, calcium carbide, nor even water gas. Benzol is not obtained from coal by passing it over a shaking screen, through a coal washer, or even by burning it in a stoker. All of these interesting and valuable chemicals are taken from coal, as well as from petroleum and natural gas, by processing—for the most part, from coal by carbonization. In this country, coal producers do not operate coke ovens, although a number of important coke producers own and operate coal mines. Nor will the coal industry do the hydrogenation, when that comes to pass. Coal producers are interested in tonnage, and are beset by problems of material handling, wage conferences, freight rates, and, if high-cost operators, in having the Government put a solid floor under selling prices. Coal producers sell the coal to coke-oven operators, who in turn sell the tar and light oils to refiners, who in turn supply to the chemical producers the various tar acids from which plastics and synthetic rubber are made. The final step is the fabricator, who molds the telephone hand set, shapes the ash tray or makes a miner's helmet.

Redman and Weil have estimated that the phenol obtained as a by-product from coking 6,800,000 tons of coal in 1941 was sufficient for the production of 8 million pounds of nylon. This would have made 256 million pairs of nylon stockings weighing half an ounce a pair. According to these investigators, on the basis of present-day operational methods, about 68 million tons of coal was required to be carbonized to supply the phenol for the estimated 1941 production of 99 million pounds of phenolic resins.

There is no immediate danger that coal will lose to petroleum and to natural gas the tonnage (in 1943 around 90,000,000 tons) that goes to by-product coke ovens, from which in turn the "coal tar products" go to the chemical industry. The ovens are built and they are operated primarily for making coke. But should the chemical industry call for vastly more benzols, phenols, and cresylics, who will supply the demand? Will the petroleum refiners, or the coke-oven operators get the new business?

Fifty years ago the first by-product coke ovens in this country were built to make a chemical, with coke a by-product, but today coke is the main product. If coal is to maintain its position as the preponderant source of raw materials for the synthetic-resins industry, which in 1940 produced 232,000 tons of plastics and for which a tonnage of 1,000,000 tons a year is easily possible, there must be a change in the economics of the industry. If through research it is found worth while to process coal to get the "coal tar products," a wider use for coke must be found. An idea, long dormant, that the by-products should be recovered before the coal, or coke, is burned for steam and power, is showing signs of new life. Coke—low, medium, or high-temperature—as smokeless fuel is another huge future outlet. Complete gasification of coke is already developed and represents another use. The use of coke in a stoker offers a smokeless fuel in automatic equipment. There is reason to believe that the yield of tar acids from coal by hydrogenation may be ten or more times greater than by carbonization, and should such a process be developed and employed there would be no coke. Coal would thus be a direct source of the chemicals required for phenolic plastics.

The situation offers great possibilities for coal, but who will do the research and develop the uses "of coal for new purposes and in ever greater quantities" as prophesied by Secretary Ickes? The coal industry has never shown aptitude for building upward to take in the consuming or processing industry. The reverse is true, and maybe that is why the phrase "captive tonnage" was coined. As long as the coal industry is content to employ its capital only for mining and transporting its product, it will not contribute to or share in the chemical industries that are now and, we hope, will be increasingly built upon its product.

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DISCUSSION

H. N. EAVENSON,* Pittsburgh, Pa.—With nearly all the conclusions in Mr. Lesher's excellent paper I am in agreement, and there is no doubt that the increased program soon to be started by Bituminous Coal Research will not solve all of the problems of the industry in regard to increasing the outlets for bituminous coal and regaining its lost markets.

This new cooperative expenditure, however, does demonstrate that the industry is becoming more research minded and that it has come a long way since the days of the coal conferences in the late twenties. I hope, and believe, that before long a way will be found by the industry to push any new developments or processes found as a result of research work, and the fact that the industry is now promoting a public relations program is an indication that this will be the case. Certainly some such way of pushing such developments is a crying need for future success.

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* Eavenson, Alford and Auchmuty.

Mr. Lesher's statement that the coal industry has been content to employ its capital only for mining and transporting coal is entirely correct; one of the main reasons is that few units in it have had enough money to promote anything but mining itself, but it is also a fact that the earnings of the units large enough to control railroads, barge lines and docks have come very largely from the branches of business other than mining. In my opinion, the successful coal companies of the future will be those following the mining of coal to its ultimate use, and while few companies can finance chemical industries, it will be entirely possible for some to help promote the development of any new products found by research, and thus to profit by the work. Such a development will be logical, and if economic conditions in the postwar period allow the coal industry to make any money so that it can attract capital, it will be probable, possibly on a cooperative basis by most companies.

C. C. Wright,* State College, Pa.—I am very sorry that Dr. Gauger was unable to be here to discuss Mr. Lesher's paper. We read it with considerable interest and I wish to congratulate Mr. Lesher on presenting such a clear-cut story.

We feel that one of the most serious problems as far as the utilization of coal is concerned is that of interesting the coming generation. We have found, in our own research organization, that it is extremely difficult to interest trained physicists, chemical engineers and chemists in doing work on coal. If they have not been brought up in an atmosphere of coal technology —either by exposure to a curriculum in fuel technology or a similar course under some other title, possibly chemical engineering—they do not have an opportunity to absorb an interest in fuels. As a result, we do not have research men and industry does not have technicians interested in fuel utilization. I think it is important that the A.I.M.E. as an organization, and particularly the Coal Division, give consideration to this matter.

Yesterday at the Coal Division meeting a resolution was passed, I believe, to the effect that this Division sponsor fellowships in mining. I think it is almost as important that we also consider the utilization angle.

S. P. Burke,* New York, N. Y.—Those of us who have been interested in research as a profession and as an industry will be particularly pleased, I think, and will recognize the fact that Mr. Lesher certainly did do a job of research on research. He has certainly gone into the subject of research and its possibilities very carefully.

I would like to add a few statistics to the general record, not because they are practical of attainment but because I think they are interesting. The coal industry at the present time is approximately a billion dollar industry in terms of its output. If that industry were to proceed, as far as research is concerned, as do those industries in the United States that devote smaller amounts of money to research, it would be putting about one per cent of that money, or one per cent of its sales value, into research. That would be ten million dollars.

If, on the other hand, it were as research minded as the chemical industry, it would be devoting about 3.5 per cent to research, or certainly something over thirty million dollars.

I am sure that Mr. Eavenson wishes there were some way to make those figures seem practical and significant to the coal industry. I recognize that such expenditures are quite impossible of attainment at the present time, but they are interesting to consider.

Another point I should like to add is that the coal industry has always had a further economic difficulty. In selling its product to the consumer the coal industry has certainly been selling—or, rather, the consumer has had to buy more freight than he buys coal. The problem of transportation of coal to the consumer has always so greatly affected the economics of the coal industry that it too has its effect on the general research problem. It seems to me, therefore, that another subject for consideration in any broad research program would be the subject of coal transportation, either in the form in which it is mined or in modified forms.

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* Professor of Fuels Technology, School of Mineral Industries, Pennsylvania State College.

* Professor of Chemical Engineering, Columbia University.

A. C. Fieldner,* Washington, D. C.—Mr. Lesher has made a timely and thoughtful contribution to the appraisal of coal research and its value to the producers and consumers of coal. His opinions deserve serious consideration because of his broad experience in dealing with coal and the coal industry. His service in the United States Geological Survey and with the John Hays Hammond Coal Commission gave him an intimate knowledge of the production of coal in all parts of the United States and of the economics of the industry. His subsequent work in the preparation of coal and in the development of new processes for the production of smokeless fuel and the recovery of by-products gives him a practical point of view in judging the value of research as a means for improving the economic status of the coal industry as well as increasing the usefulness of the industry in providing better and more valuable products.

Until comparatively recent years, most of the coal research of this country was conducted by the Federal Bureau of Mines, certain state geological survey and university laboratories, and a few industrial and endowed research organizations. Only a very few coal operating companies conducted any technical research worthy of the name. Naturally, much of the coal research of former years was concerned with the occurrence, composition, and properties of coals and efficiency in utilization rather than the immediate commercial problems of the coal producer in maintaining his markets and manufacturing new products from coal. Such research needs the direct support of the coal industry. It should be conducted both by the individual companies and by association of coal and fuel-equipment companies in a common effort.

Fortunately, this new phase in coal research has now arrived in this country as well as in Great Britain. Bituminous Coal Research, Inc., has embarked on a $2,500,000 program covering a period of five years supported by yearly subscriptions from the bituminous coal industry. The anthracite operators likewise have an important research program in progress, and in Great Britain the British Coal Utilization Research Association has announced that its total funds obtained from Government and industry for the past year exceeded 200,000 pounds. Great Britain is spending considerably more money for research per ton of coal mined than is the United States.

I am glad to see this rapidly growing interest of the industry in supporting its own research. It should ensure proper attention to practical problems, which will give more or less immediate results. Industry also should give financial support to fundamental scientific research pertaining to coal and its products at universities and research institutes. The schools, especially, need support to train men for coal research.

Governmental agencies, state and national, probably will continue to do considerable coal research of a fundamental and long-range nature. Much of it will relate to conservation of men and resources in the coal industry. Some competition in research will occur between different agencies, and this is not without advantage in spurring on the workers in this great field of endeavor.

J. E. Tobey,* Fairmont, W. Va.—The bituminous coal industry surrendered its birthright to become a manufacturing industry during the transition from the beehive to the by-product coke oven. (There are probably good and sufficient reasons for this situation.) The petroleum industry chose the other path, and it has become a great manufacturing industry as well as a producer of basic raw material. Thus the pattern has been set.

The petroleum industry will manufacture oils and gasoline from coal. The chemical industry will manufacture synthetic rubber, plastics and tens of thousands of new compounds from coal. As plastics encroach on the fields of steel, tin, paper, wood, glass and textiles, this group of manufacturing industries will probably engage in the rolling, pressing, hammering, casting, extruding and weaving of this newcomer. The steel industry and merchant plants will continue to manufacture coke from coal.

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* Chief, Fuels and Explosives Service, U. S. Bureau of Mines.

* Director, Coal Bureau, Upper Monongahela Valley Association.

Now the question is: Where does this leave the coal industry? The answer is obvious. In the present cycle, the coal industry will remain a basic raw-materials industry. There are, however, two important trends, which will affect it physically:

1. Increase in captive coal reserves. (The manufacturing industries that use coal by-products will undoubtedly safeguard their sources of raw material by the purchase or long lease of coal reserves.)

2. Increase in the corporate size or management of coal holdings in noncaptive mines.

It is interesting and significant to note that the trend is definitely downward for captive holdings by concerns such as utilities and manufacturers whose principal use of coal is for fuel purposes. Thus the differentiation—the captive tonnage trend is upward for by-product coals and downward for fuel coals.

An appraisal such as this is necessary before the matter of research can be properly considered for the coal industry. From this appraisal, the following deductions can be made:

1. Cooperative research by the coal industry has definitely received the green light to proceed on work to improve the fuel uses of coal. This is its primary function, and it should include the closest collaboration of equipment manufacturers.

Fuel uses is a large field, embracing residential heating, steam generation and industrial processes, carbonization and gasification.

2. Fundamental and industrial research should both be carried on in the interest of fuel uses.

3. Fundamental research should be employed to "spark" the development of new nonfuel uses for coal and its findings should be passed on to the manufacturing industries.

4. Wherever coal utilization is retarded by the existence of a "no man's land," the coal industry should do research work in that area.

5. Greater than the need for research will always be the need for mass application of the findings of research.

P. D. Foote,* Pittsburgh, Pa.—The excellent summary of Research for the Coal Industry by C. E. Lesher seems to me a pathetic record and a most serious indictment of the industry and its officials. No wonder that the various financial advisory agencies are recommending this temporary boom period for the unloading of coal securities, the taking of losses, and conversion of what remains to the disappointed investors, into securities of corporations showing some initiative and responsibility for the future. Seven hundred thousand dollars for the entire coal research of this nation, of which more than half comes from the Government and a good part of the rest from industries unrelated to coal—these are astonishing statements! The research expenditure of all agencies combined amounts to one tenth of a cent per ton of mined coal.

Many progressive chemical companies spend 5 per cent of gross sales on chemical research. We may thank Heaven that the petroleum interests have not followed the precedent of the coal people, for had this been the case, the current war might have been terminated with Americans as slaves. The petroleum industry has been managed by officials who appreciate their responsibility to the nation, to their employees, and to their corporate stockholders, and who realize that this is an age of scientific development requiring intensive research to keep abreast of the times. Many small petroleum companies spend more for research than the entire coal industry, and the larger petroleum corporations are dealing in figures which to the coal man must appear astronomical.

The coal industry should mean more than a mere mining and excavation enterprise. On the same basis the petroleum industry would be content with bringing crude oil to the surface of the ground and would hand over the intricate problems of modern refining, transportation, marketing, and the like to independent agencies.

Coal, to the probable astonishment of the coal-industry executive, is almost on the threshold of a brilliant future in this nation's history. Five cents per ton of coal mined would mean 33 million dollars a year for research. In view of the accumulated deficit of fundamental technical knowledge, this is not an unreasonable sum to devote to coal research. The money should be continued over a period of at least fifteen years before major results could be expected. The work would lead the industry into chemical and engineering enterprises and even into a moderate tonnage of chemical by-products, of which aspirin might be a good starting point for many of the present officials.

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* Gulf Research and Development Corporation. Discussion reprinted from *Min. and Met.* (June 1944).

Fortunately we do have in this country industrial leaders of vision, who feel a responsibility to the public, to the nation, to employees, and to investors, and these men will take the coming play away from the coal industry, allowing the latter to continue, as at present, in its capacity as underground excavators groping in the dark.

What the coal industry needs are leaders and corporate officials who recognize the potential value of research and are willing to spend money for long-range development.

Discussion on Hydrogenation of Coal and Lignite (*Continued from page* 362)

I have wondered why it would not be feasible to produce water gas from a Lurgi char, and not use the Bergius process. It seems to me more economical. I wonder if Dr. Fieldner agrees with that theory?

A. C. Fieldner.—The Lurgi process is used in Germany to carbonize brown coal. The char from that process is used for making water gas for the Fischer-Tropsch process. In some of the German plants the low-temperature tar goes to the Bergius plants for hydrogenation and the char gas to the Fischer-Tropsch plants for synthetic fuels. Is that correct, Mr. Cooke?

M. B. Cooke.—I believe that is correct and looks like a balanced operation to me.

A. C. Fieldner.—That is feasible.

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